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UNDER CURRENTS

NORTH BLOOMFIELD HYDRAULIC MINE, NEVADA COUNTY, CAL.

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PRACTICAL GOLD-MINING:

A COMPREHENSIVE TREATISE ON THE
ORIGIN AND OCCURRENCE OF GOLD-BEARING
GRAVELS, ROCKS, AND ORES, AND THE
METHODS BY WHICH THE GOLD
IS EXTRACTED.

BY

C. G. WARNFORD LOCK,

CO-AUTHOR OF 'GOLD : ITS OCCURRENCE AND EXTRACTION.'

WITH 8 PLATES AND 271 ENGRAVINGS IN THE TEXT.



E & F. N. SPON, 125, STRAND, LONDON.
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1889.

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INTRODUCTION.

AN American writer recently gave us a sketch of what California would have been by this time, supposing the great gold discoveries there had never taken place. But who can realise how much the whole world would have been thrown back in civilisation, in colonisation, and in commerce, but for those lucky finds in California and Australia some forty years ago? Every nation in history, and every member of each nation, has shown the same desire to accumulate the precious metal. The all-conquering Roman drove his slaves and prisoners by regiments to toil in the gold mines of Hungary, and miles of tunnel there may yet be seen carved in the solid porphyry by hammer and chisel, the mines being worked to this day. What the ancient Roman did by forced labour the Anglo-Saxon does now by the aid of steam, and explosives, and hydraulic power. All the chief gold mines of the world, excepting only the Siberian, are controlled by English-speaking peoples, and the metal is coined principally in British, Colonial, and American mints. Gold mining now employs millions of pounds of capital and millions of workmen, and has grown from a precarious calling followed by the few, to a recognised industry conducted on scientific principles and on a gigantic scale. There is no longer any excuse for the fulfilment of the old Portuguese proverb—"a silver mine brings wretchedness, a gold mine ruin,"—if people would only exercise ordinary common sense, and take the trouble to learn by the example of others. No other industry can rest on such a sound basis, for the simple reason that the product is always marketable and never varies in value. Want of success in working a gold mine is generally due to one cause (barring accidents and fraud), viz. *ignorance how to extract the gold*. To convey information which will help to dissipate that ignorance, this volume has been written. The superficial reader will probably see in the book a family likeness to

its predecessor—"Gold : its Occurrence and Extraction,"—produced by the present author and his father in 1882, and now out of print ; but though much of the matter in the earlier work has been retained, for the sufficient reason that its utility and value have not diminished, an effort has been made to supplement it with the most recent facts and figures obtainable. And if no great discovery has marked the interval between the two books, the seven years' experience gained has developed many new features, which no one interested in the industry can afford to neglect.

In conclusion, the author will be extremely thankful to receive at all times notices of new processes and machinery for publication in subsequent editions, and will gladly correspond with Directors and others who are encountering difficulties in the development of their property.

C. G. WARNFORD LOCK.

ARPAD GOLD SYNDICATE,
20, MOORGATE STREET,
LONDON, E.C.

May 1st, 1889.

SYNOPSIS OF CONTENTS.

CHAPTER I.

GEOLOGICAL OCCURRENCE OF GOLD DEPOSITS, AND MINERALOGICAL ASSOCIATIONS OF THE GOLD, pp. 1 to 107.

This chapter embraces three separate subjects :—(1) A summary of the theories advanced during the last 30 years to account for the origin and formation of gold in veins and alluvial deposits, and for the occurrence of nuggets ; (2) a systematic epitome of the geological formations in which gold has been found, showing its very wide distribution in strata of almost all ages ; (3) a catalogue of the metallic and other minerals associated with gold and auriferous rocks, with remarks upon the manner in which the ores are affected by them.

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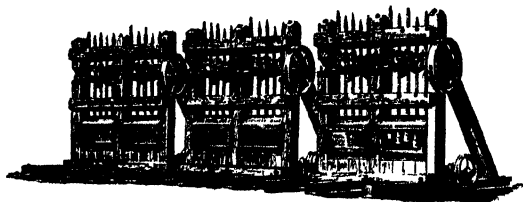
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MINING MACHINERY.

GOLD and SILVER.

Stonebreakers, Stamps, High-speed Rolls, Chilian Mills, Conveyors, Elevators, Frue Vanners, Blanket Strakes, Drying and Roasting Furnaces, Amalgamating Pans, Settlers, Barrels, Retorts, Patent Chlorination and Leaching Plant.

COPPER, LEAD, ZINC, &c.

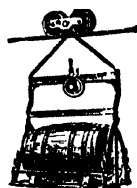
Stonebreakers, Crushing Rolls, Trommels, Elevators, Classifiers, Jiggers, Patent Buddles, Percussion Tables, Reverberatory and Water Jacket Furnaces, Blowers, Slag Pots, Moulds, &c.

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"For Aerial Tramway."

COMMANS & CO.,
52, GRACECHURCH STREET, LONDON, E.C.

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PRACTICAL GOLD-MINING.

CHAPTER I.

GEOLOGICAL OCCURRENCE OF GOLD DEPOSITS AND MINERALOGICAL ASSOCIATIONS OF THE GOLD.

IT is probably no exaggeration to say that preconceived notions and hasty theories concerning the formation and geological age of gold have done more to retard the gold-mining industry than to advance it, and that many of the most important discoveries of recent years have been in direct opposition to the dicta of the greatest authorities. This, indeed, is hardly surprising when we reflect what a very small proportion of the earth's crust has ever been examined at all, and observe the absence of uniformity among geologists even in the names bestowed upon the rocks that have been examined. In the belief that our knowledge is still quite inadequate to account satisfactorily for the manner in which metalliferous veins are formed, and to define the laws which govern their formation, no attempt will be made in this volume to promulgate any new theory on the subject, nor to give precedence to any one in particular of the existing theories. The half-dozen most recent opinions of men best entitled to discuss the question will be concisely stated, so far as they deal especially with gold; and will be followed by a full yet simple categorical statement of the geological formations in which gold has been found, incorporating all details of scientific or industrial value; as well as with a similar account of the various mineral associates of gold, especially with a view to elucidating the character of the association. It is believed that by thus confining remarks to a clear arrangement of ascertained facts, this chapter will be found much more valuable for purposes of reference, and may form a foundation on which to build other facts as recorded.

ORIGIN AND FORMATION.

Rosales on Origin of Veins.

The first notable recent attempt to explain the origin of auriferous quartz veins was Henry Rosales's prize-essay, written for the Victorian Government, in 1860. It is substantially as follows :—

“Auriferous quartz lodes are unlike most other lodes, which are

'crevices more or less vertical, caused by contraction during drying, or by metamorphism, or by mechanical disturbance of a rock, this crevice having been subsequently filled up.' They are in their origin anterior to all those forces, some of which accompanied the eruption of granitic rocks, and which have been thoughtlessly applied by some to explain the origin of quartz lodes. The fact that the quartz lodes are of an earlier date than the granite, forces a further investigation of the subject to a remote period of the earth's history, when the granitic rocks not yet having made their appearance, the Cambro-Silurian beds were still undisturbed in their original horizontal position. The Cambro-Silurian system of Australia presents a series of coarse- and fine-grained sandstone, containing few marks of slaty structure; slaty sandstone of different colours, alternating with bands of slate of perfect cleavage, also of different colours, but generally exhibiting a greenish hue, and white when decomposed. Organic remains seem to be of rare occurrence in this formation. This far and widely spread Palæozoic series of rocks—the waste and refuse of the primitive cooling crust of the earth's surface, was deposited slowly, gradually, and without interruption in horizontal beds, which thus attained the enormous thickness they now present during that protracted period, when peculiar cosmic and telluric agencies, all as yet singularly averse to organic life, were at work. While, however, there are no apparent signs of mechanical disturbances during the long period that elapsed from the cooling of the earth's surface to the deposition of the Silurian and Cambrian systems, it is to be presumed that the internal igneous activity of the earth's crust was in full force, so that on the inner side of it, in obedience to the laws of specific gravity, chemical attraction, and centrifugal force, a great segregation of silica in a molten state took place. This molten silica continually accumulating, spreading and pressing against the horizontal Cambro-Silurian beds during a long period, at length forced its way through the superincumbent strata in all directions; and it is abundantly evident, under the conditions of this force and the resistance offered to its action, that the line it would and must choose would be along any continuous and slightly inclined diagonal, at times crossing the strata of the schists, though generally preferring to develop itself and egress between the cleavage planes and dividing seams of the different schistose beds. Thus were formed in a more or less horizontal position, in all directions, innumerable flakes and extensive sheets of quartz rock, apparently interstratifications as regards their strike, but only apparently such, for they distinctly traverse and intersect the underlie of the slate rocks, being thicker between the schistose planes, and narrower when intersecting them. From the quartz rock started quartz veins, some (β) running almost parallel with, and others (γ) perpendicular to its position, while

other veins (δ) shot out in capricious planes and directions. These veins or leaders all thin and run out in comparatively short distances, especially such veins as cut across the slates at the line of the greatest resistance. Simultaneously with the upheaval by the granitic rocks of the Cambro-Silurian slates in average meridional line, the approximately horizontal main quartz belts were upheaved and placed on edge along with the schist strata by this general disturbance. It is in consequence of this change that the quartz belts are apparently interstratified, while, in reality, they are merely intersecting. The positions having been altered, what formerly was or approached the horizontal, became perpendicular, and *vice versa*, so that horizontal sheets of quartz reefs which had been forced between the schistose cleavages and different strata, appeared as almost perpendicular quartz lodes, their strike being conformable to the general meridional bearing of the schist, and the coincident line of upheaval, their underlie intersecting that of the slates to the east or to the west, the veins (β) which ran parallel, following the main course of the quartz rock, the perpendicular veins (γ) becoming horizontal or flat, while the other veins (δ) would take their respective analogous positions. In the same manner, the horizontal sheets of quartz rock, when upheaved in the medial line of action, would show along their approximate meridional line a varying shoot to the north, or to the south; a north-east or north-west horizontal development of quartz rock would thus necessarily, when upheaved, have its shoot northerly, while a south-east or south-west stretch would present a southerly shoot, and a lateral upheaval would, of course, reverse the above order. In quartz lodes where there is no noticeable or well-defined shoot in either direction, it may be inferred that their original development was indifferently either north, south, east, or west; and this is precisely the appearance which auriferous quartz lodes present in nature to the miner and geologist; they form innumerable more or less perpendicular quartz dykes and extensive quartz rock belts which strike, but with few exceptions, in an approximate meridional line, thus disclosing to view, on a gigantic scale, that remarkable parallelism which, after all, is but a natural feature necessarily consequent on the almost unvarying strike above alluded to. These auriferous quartz lodes intersect the strata of the slate rocks, and are cased with walls of slate and sandstone; they have quartz veins issuing from them in various directions across the country as leaders, flat veins, &c. They sometimes form themselves into irregular masses of veins, at other times they appear as massive bodies of quartz rock which dwindle into strings that serve as the connecting links with some other quartz blocks. These facts go to show that the quartz lodes when forcing their egress often disturbed, fissured, and rent the enclosing schists, the openings so

effected being instantly filled by the quartz stone, thus giving rise to those capricious irregular or zigzag shapes vulgarly termed east and west veins, &c., which are frequently met with in underground workings.

"But there are other than the cosmic and geological conditions mentioned which prevailed at the time of the origin of quartz lodes, and they also equally indicate the plutonic character of this dyke formation. Under this head is to be reckoned the occurrence of felspar in quartz veins, for it is an established scientific fact that mica, felspar, and amphibole or augite, are all minerals, none of which can be formed apart from igneo-chemical action. This single fact would alone go far to indicate the originally molten state of the silica of quartz lodes. Another argument to the same end may be drawn from the fact that the auriferous quartz lodes have exercised a manifest metamorphic action on the adjacent walls or casing; they have done so partly in a mineralogical sense, but generally there has been a metamorphic alteration of the rock. Hence it is that in the immediate contact of the quartz lodes the schist or fragments of it are generally more or less micaceous or altered into their laminæ of mica, crystalline laminæ of nacrit or of chlorite, which has invariably tinged the adjoining quartz with a green colour. Among these minerals is one at times disintegrated which shows the cleavage of orthoclase. There are but few minerals found at the contact of the schist and quartz rocks. This, however, is only natural, for the interchanging rocks were of simple chemical composition. The metamorphic influence exercised by the quartz rocks on the bordering strata is very striking, though it is not easy to distinguish it all over them; it is pre-eminent in the mining district of Tarrangower, where all the quartz lodes are separately checked and walled by distinct accompanying strata of dark siliceous schist or Lydian stone, evidently slate and sandstone schists hardened by the metamorphic action of the quartz lodes, in the same manner as when acted on by igneous or volcanic rocks, apparently changing the physical conditions without altering the chemical quantities. This metamorphic action is observable in the Tarrangower district for many miles. In the Bendigo district, the metamorphic action is also to be seen, quartz rock belts being often carried in between hard ferruginous schists which, however, are generally disintegrated, and do not therefore present any very prominent metamorphic features. In the Ballarat district, the metamorphic action of the quartz lodes may also be detected, although the rocks there are even more disintegrated than at Bendigo; it is not very easy, therefore, to distinguish their mineralogical composition with any tolerable degree of accuracy. The hard, compact, partially disintegrated strata of Ballarat, which separate the quartz belts from each other, may be considered to correspond geologically with the metamorphic siliceous slate of Tarrangower. The same metamorphic

action can be traced throughout the districts of Amherst, Avoca, Creswick, &c. Another reason for the igneous origin of the auriferous quartz lodes consists in the mechanical disturbance caused by the protrusion of the quartz lodes themselves. It is to be constantly observed that the strata of the schistose rocks are more or less contorted—that their underlie is variable to the east or to the west—that fragments of them are entangled in and metamorphosed by the quartz lodes, and that in consequence the adjacent ‘country,’ to use a mining expression, is frequently broken up into fragments, forming a breccia of commingled slate and sandstone, whilst the quartz lodes and veins traverse the same country intact and unbroken. The inevitable inference is that the disturbance was caused by the forcible protrusion of the quartz lodes. From the above facts and deductions, it can therefore be maintained that the gangue of the auriferous quartz lodes is of igneous origin, and not the result of the gradual deposition of quartz from a siliceous solution. In addition to the positive arguments already adduced, which base themselves on geological observations, there are also negative ones, the result of scientific deductions, which lead to the same conclusion. Starting from the established fact that silica is soluble in water and watery vapour, during its separation from alkalies, the first considerations which present themselves are in reference to the supply of the great quantity of alkaline silicates, which at that time could only be of a felspathic nature. How under the then existing circumstances could the enormous quantities of water or water-vapours, requisite to dissolve the immense mass of quartz which was to form the innumerable quartz lodes now existing, continuously find their way into the bowels of the earth? The number of these quartz lodes could hardly be attributed to aqueous agency, as a solution contains no intrinsic forcing power, and could therefore have but comparatively few outlets. More than this, the felspathic rocks having been deprived of their alkaline silicates to form silica, there would remain an immense bulk of clay, and the disposal of this residuary mass is not accounted for in any way. It is another weighty consideration also, that the formation of quartz lodes by the deposition of silica, from an aqueous solution, would necessarily involve the production of hydrous silicates, zeolites, hyalite, opal, &c., which are always present in the deposits made by siliceous thermal springs, as in the ‘geysirs’ in Iceland, and indeed in all volcanic eruptions in which there were aqueous vapours: it is so in basalt, &c. These are all minerals which authenticate the presence of water-vapours at the time of their formation, and are to be found in the gangue of many metalliferous lodes, as at Andreasberg; Strontian, in Scotland; Ciclowa, near Oravicza, in the Bannat; Huelgoet, in Brittany; Kongsberg, in Norway, &c. The quartz lodes of this country show a character quite distinct from that just now referred to;

for nowhere are the hydrous silicates, or the other minerals named, to be detected in them; and it is therefore to be inferred that their non-existence demonstrates the impossibility of the origin of quartz lodes being due to an aqueous solution.

"The next question is as to the metalliferous character of the quartz lodes. There are but few metals or metallic minerals to be found in them: they are gold, iron, and arsenic pyrites, the last two in some instances in great quantities; copper-pyrites, zinc-blende, galena, molybdenite, pharmacosiderite, hematite or glaskopf, and malachite; the last three, however, are oxygenated, and are therefore to be looked on only as minerals of secondary formation, the result of the disintegration of the primitive minerals, which are arseniurets and sulphurets. These minerals ascended simultaneously with the quartz; and the contemporaneous formation of the quartz gangue, arseniurets, and sulphurets implies the forcing up of these minerals in a sublimated state. The heat of the molten silica would necessarily volatilise the gold as a vapour of purple colour, and would also sublime the arseniurets and sulphurets, which are all volatilisable without being decomposed, at a much less temperature than gold, air being excluded. They are found decomposed only near the surface. Thus the purple fumes of metallic gold and sublimated vapours of the arseniurets and sulphurets of other metals, entering the quartz gangue, permeated it as gaseous vapours, forming veins, shoots and streaks, interlacing the gangue in the direction of its stretch, penetrating also into the recesses of the quartz veins and leaders, the gold being precipitated in gold-leaves, film, &c., on comparatively cold bodies, such as the sides of the lodes, or entangled pieces of schist, and accompanied by the sulphurets and arseniurets. The sulphurets, arseniurets, and other volatile metals influenced the volatilisation of gold, and in this way it was carried into and lodged in the crevices, joints, and sides of the lodes, where it could not have reached unless accompanied by the sulphurets and arseniurets. Hence we find gold in its metallic state mixed mechanically with iron- and arsenical pyrites; sometimes it is even perceptible to the naked eye, at other times it is not; and it is also found mixed with galena and zinc-blende. Indeed it is scarcely possible to find either of these two minerals without at the same time finding gold in contact with them. Of the minerals enumerated, iron- and arsenical pyrites are found in large quantities in the quartz gangue; but copper-pyrites, galena, and zinc-blende, are seldom found, and then in insignificant quantities. It is to be remarked that the affinity of these different minerals is according to the following scale—first, galena, then and almost if not quite equally, zinc-blende, arsenical pyrites comes next, and iron-pyrites follows; and therefore gold is contained in the gangue along the shoot of these metalliferous ores; but where the gangue is far apart

from the metalliferous indications, it is generally barren. On this theory, it is not difficult to account for the flat leaders and the running out of the quartz stretches, now the caps of the reefs, being often richer than the rest of the gangue, or to account for the poverty or richness of reefs, when they suddenly become contracted for a certain length, conformable to the greater or less opportunity offered to the gold to precipitate according to the physical circumstances, such as mechanical impediments, change of temperature, &c.

"It has been attempted to explain the origin of gold as being the result of precipitation by iron from its solution, under the influence of electricity. Without entering into the chemical part of this theory, it may be sufficient to remark that the theory in question is one-sided, only accounting for the formation of gold, but not of the sulphurets which would be decomposed. Strictly, then, the simultaneous deposition of gold, sulphurets, and arseniurets, goes to prove chiefly that a very feeble electrical agency was at work in the formation of the quartz lodes. However, it was owing to the subsequent electrical influence, at first an electro-chemical one evolved by the disintegration of the primitive minerals, that even remote substances were decomposed, and that new combinations were formed—the secondary minerals, such as cube ore, pseudomorphous hematite, &c.; their elements being transferred by electrical currents even through moist non-conducting solids, and in some cases during the process they were deprived of their chemical properties, or in other ways influenced by electric agency. It is to the action of these currents of electricity, whose intrication Fox has so well described, that the present partially or entirely disintegrated state of the upper levels of the quartz lodes is chiefly to be attributed. From the igneous origin of auriferous quartz lodes and metallic ores, it would be correctly inferred that the gangue would exhibit a homogeneous character; this, however, is not always the case, although it is so generally. Where it is not so, it is no doubt due to the subsequent reopening of the gangue fissures, more recent protrusions of quartz rock having almost disconnected the more ancient veins, or else formed a new body in their midst, and so giving to the whole, for some distance, a flaky, laminated, or seamy appearance, which is greatly increased by entangled schistose fragments and veins of metallic substances, either decomposed or otherwise. But that order of deposition of different substances, corresponding with the faithful parallelism from the sides of the lode towards its centre, cannot be found, though it is a remarkable feature in many veins containing carbonates of lime, iron, &c. Werner first called attention to this striking arrangement, of which Weissenbach has given numerous examples, and which was evidently caused by the agency of water. Thence it is that the seamy appearance in some portion of the quartz lodes does not point

to the conclusion of an aqueous origin, for it is irregular, however apparently otherwise at first sight, as it does not fulfil the above-described conditions of parallelism. In some districts the quartz lodes have been disturbed by felspathic or igneous rocks, forming dykes of a more recent epoch. Hitherto the felspathic dykes of Bendigo and Maryborough have received the most attention; in the former district, they are found at times traversing the quartz lodes, following their strike, or faulting them; and in other cases, separating their course from them, they crop out on the surface. They are greatly disintegrated, so much so that it is not possible at present to give their mineralogical composition with accuracy, or to determine their geological age beyond what has been already advanced. They do not seem to contain zeolites, but appear to be intimately connected with the subsequent changes in quartz lodes, such as the re-opening of the quartz veins, and also the occurrence of large masses of arsenical pyrites, which, and it is a most interesting fact, as it shows the connection of this metalliferous ore with heat, contain a felspathic mineral, as already stated. In the latter district, they have been called quartz-porphry, and seem to be of precisely the same character as those of Bendigo; they disturb and are connected with the quartz lodes in a similar manner, and are in a state of partial disintegration; their mineralogical character may be somewhat different, as their name—quartz-porphry—indicates. It is, nevertheless, probable that both these igneous rocks are contemporaneous. At Ballarat no igneous dykes have been yet observed, either on the surface or underground; thus disturbances as 'faults' are of rare occurrence; up to the present time, none of any consequence have been found, and it is a remarkable fact, also, that arsenical pyrites has not been found there, either in small or large quantities, as at Bendigo, Tarrangower, &c. In the sandstone walls of a very few quartz lodes, the empty impressions of arseniuret of iron have been found. Would the absence of igneous dykes not seem to presuppose that the quartz lodes have not been reopened, and that, therefore, arseniureted masses of auriferous character could not have been injected? And might not the comparative poverty of the Ballarat quartz lodes be also thence presumed? From what has been stated as to the theory of the igneous origin of auriferous quartz lodes, it may be logically deduced that the presence of sulphurets and arseniurets in a quartz lode is an empirical test of its comparative auriferous character; that the appearance of igneous dykes in connection with quartz lodes, and contemporaneous with those mentioned, would give a more auriferous stamp to any district; and that auriferous quartz lodes are to be expected, intersecting the schistose formation, at any depth. The last statement is borne out by the following considerations: first, it can be easily imagined, if the enormous abrasion be taken into account, how deep the present

surface with its yet auriferous quartz lodes, must have been under that which existed before abrasion took place; again, the granite which simultaneously upheaved the Cambro-Silurian and auriferous quartz rocks, disturbed the thick schistose formation with such gigantic force that it contorted and placed the beds on edge, thereby causing enormous convulsions and faults, on a scale too grand to be noticed by the miner; and it is evident that in some instances strata of the schistose and quartzose formation which, before the upheaval of the granite, were far below the formerly existing surface, have been disclosed by that upheaval, and that these strata contain quartz lodes in no wise less auriferous. It may, therefore, be maintained that to as great a depth as the quartz miner can ever penetrate, he will find auriferous quartz lodes, and that deeper still there are others equally auriferous."

Another view of the question is taken by Richard Daintree, in his report on the geology of the district of Ballan, Victoria, in 1866, when discussing whether the drifts underlying the old amygdaloids of Bacchus Marsh are theoretically likely to contain drift-gold. The problems proposed for solution were, "(1) the age and origin of quartz reefs, (2) and whether the gold is contemporaneous with their formation. To answer the first question, in regard to age, he sought sections where strata of determined age, containing quartz reefs, are seen overlain unconformably by other sedimentary strata of known age. Such are to be found in the Werribee gorge, where the quartz strings and reefs traversing Silurian slates, &c., cease abruptly at the junction of the 'Lower Mesozoic,' from which may be inferred that these quartz reefs at least were formed prior to the superincumbent strata. In Gippsland, again, Upper Devonian conglomerates, with their associated claystones and sandstones, enclosing *Lepidodendra*, &c., are barren of quartz reefs, while the Silurian on which they rest have reefs well defined." He then quotes Hartt, on the "Gold of Nova Scotia of pre-Carboniferous age," as follows:—

"At Corbitt's Mills, about 4 miles north of Gay's river, Colchester county, Nova Scotia, auriferous clay-slates of the same character as those of the other gold-districts of the province, are overlain unconformably by nearly horizontal beds of grey and red conglomerate, grit, and sandstone, of Lower Carboniferous (probably lower coal measures) age. At the mills, these last are only a few feet in thickness. They in turn are overlaid by a mass of drift, and by beds of stratified sand and clay of variable thickness. As to the Carboniferous age of the conglomerate and sandstones there can be no doubt. They cannot be Silurian, for they overlies unconformably rocks of this age. They are totally unlike any Devonian rocks occurring in the province, while they agree perfectly with the Lower Carboniferous conglomerates and sandstones of the Carboniferous basin on the margin of which they lie. They

contain a few ill-preserved fossil plants like those found in similar Carboniferous beds. Between the Carboniferous and the Drift the only formation occurring in Nova Scotia is the New Red Sandstone, to the rocks of which the beds under consideration bear no resemblance. They cannot be of Drift age, for their fragments form rounded boulders in that deposit. They show no signs of having suffered metamorphism. The lower part of the beds of conglomerate or grit, at their junction with the slates, is richly auriferous, the gold occurring principally in the form of flattened scales, sometimes $\frac{1}{4}$ in. in diameter, disseminated through the rock. I have seen many fragments of the conglomerate, not 1 cub. in. in size, on the surface of which 20 or 30 scales of gold could be counted with the naked eye. Gold has been washed from the drift overlying the conglomerate. The source whence the gold was derived was doubtless quartz veins in the clay-slates. Only one vein, about $\frac{1}{4}$ in. thick, has been discovered beneath the conglomerate. It is richly auriferous, and has a strike of about north and south, and a dip of 70° east. Non-auriferous quartz veins are very numerous in the slate hills of the vicinity. That this vein is older than the Carboniferous strata is plain from its ending abruptly at the junction with the slates. From the above facts, I think there can be no doubt that the gold of Corbitt's Mills is of pre-Carboniferous origin, and since the gold of that locality was derived from strata precisely similar in character to those of the other gold-regions of Nova Scotia, and which strata are but the reappearance northward of the gold-bearing rocks of the gold-fields of Renfrew and Oldham, and of the metamorphic band of the Atlantic coast, I think that the pre-Carboniferous age of the gold of Nova Scotia is clearly indicated. It is a very generally accepted theory, propounded by Sir Roderick Murchison, that whilst gold is confined to Lower Silurian strata, it did not make its appearance therein until just before the time of the Drift. As the gold of Nova Scotia was probably introduced into or assumed its present form in the quartz veins at the time of the metamorphism of the Silurian rocks, which metamorphism was pre-Carboniferous, I have doubted the correctness of this theory. The occurrence of gold in the Carboniferous rocks of Corbitt's Mill shows that it is not to be applied to the province of Nova Scotia.'

"On Keelbottom Creek and Star river, tributaries of the Burdekin river, in the district of North Kennedy, Queensland, is a great thickness of Devonian rocks, resting on, and no doubt the cause of metamorphism in, underlying gold-bearing gneiss, mica-schists, and hornblende slates. These are associated with very thick beds of quartz rock, rarely with quartz reefs; though traversed with innumerable thin strings and veins of quartz, such reefs and veins never pass into the Devonian series. Although the gold is most abundant in the loose drift resting on the

metamorphic rocks, still, where Devonian conglomerates occur, more or less alluvial gold is found in their débris, even where an outcrop of metamorphic rocks is many miles distant. The geological survey of California has ascertained 'that a large portion of the auriferous rocks of California consist of metamorphic Triassic and Jurassic strata.' It is evident that the reefs in these rocks are more recent than those of Hartt in Nova Scotia, or than those of the Werribee gorge, or the Upper Burdekin.

"No subject has perhaps been more discussed than the formation of mineral veins; laying claim to no original ideas on the subject, I shall simply bring forward the published statements, which most coincide with the results I have arrived at by independent observation. I had long ago come to the conclusion that most, if not all, the gold in the quartz reefs was derived from the rocks in which these reefs occur. That the strata themselves received their supply of gold at the period of their deposition from the ocean in which they were deposited. That organic matter, and the gases generated therefrom on decomposition, sulphuretted hydrogen, &c., were the cause of the precipitation; and that the amount of metallic deposit was in proportion to the amount of organic matter deposited with the oceanic sediment. That subsequent plication and desiccation of the sediment caused fissures, into which the mineral waters percolating the boundary rocks flowed and were decomposed, and their mineral contents were precipitated, possibly by magnetic currents, thus causing mineral veins.

"Sir W. Logan says:—

'The observations among the gold-bearing rocks of the Southern States seem to show that the precious metal was originally deposited in the beds of various sedimentary rocks, such as slates, quartzites, and limestones; and that by a subsequent process it has been, in some instances, accumulated in the veins which intersect these rocks. The formation of these veins would seem to be subsequent to the Silurian period.'

"Again, T. Sterry Hunt says:—

'The reducing power of organic matter is further shown in the formation of metallic sulphurets, the reduction of sulphates having precipitated in this insoluble form, the heavy metals—copper, lead, and zinc—which, with iron, appear to have been in solution in the waters of early times; but are now by this means also abstracted from circulation, and accumulated in beds and fahlbands, or by a subsequent process have been re-dissolved and deposited in veins. All analogies lead us to the conclusion, that the primeval condition of the metals and of sulphur was like that of carbon, one of oxidation; and that the vegetable life has been the sole medium of their reduction.'

“And as a corollary to the same ideas, expressed in different words, the same author says :—

‘The intervention of intense heat, sublimation, and similar hypotheses, to explain the origin of metallic ores, we conceive to be uncalled for. The solvent powers of solutions of alkaline carbonates, chlorides, and sulphurets, at elevated temperatures, taken in connection with the notions above enunciated, and with De Senarmont’s and Daubrée’s beautiful experiments on the crystallisation of certain mineral species in the moist way, will suffice to form the basis of a satisfactory theory of metallic deposits.’

“Having now considered the age and origin of quartz reefs, let us turn to the other question of contemporaneity of reefs, and their associated gold. Let us first consider if solution and re-precipitation of gold is still going on. We can then better form conclusions on this subject. In testing a solid mass of iron-pyrites, given me by the Director of the Geological Survey, gold was found throughout. The mass retained the structure of a tree-stem, and was a replacement of the organic structure by pyrites, and had been taken from the Ballarat drift. The same experiment on another tree-stem, taken from the same drift, was repeated by Newbery, the Geological Survey analyst, with a like result. Unless this gold was carried in a soluble form into the pores of the wood, and there precipitated with the pyrites, it would be difficult to account for its presence. Foord, the well-known chemist and metallurgist of Melbourne, has repeatedly informed me that, in operating on the St. Arnaud silver-ores with hyposulphite of soda, and precipitating the dissolved metals from the solution, he had usually found that an appreciable amount of gold had been dissolved with the silver, indicating that in that mine, at least, gold may exist as an ore.

“Whether this fact points to the real solvent of the gold and silver precipitates, from their first storehouses in the sediments to their accumulation and re-precipitation as metal in the reefs, is worthy of consideration ; certain it is, that where found in the sediments themselves, the noble metals are usually associated with sulphur compounds, iron-pyrites, &c., and there is no reason why hyposulphites of the alkalis should not be formed in the mineral waters percolating these strata. At the same time, it is possible the St. Arnaud case may indicate more the possibility of gold as a chloride combined with chlorobromide of silver.

“In whatever direction we look for the cause of the original precipitation and re-solution for after-deposit in mineral veins, we must never lose sight of the fact that the first agent must have been potent to precipitate both gold and silver, and the second to re-dissolve the united precipitates, as no gold has yet been found in nature unalloyed with silver. That sulphur compounds have played an important part in the reactions

which we have endeavoured to explain is evidenced by the fact that scarcely ever has pyrites taken from the Silurian slates of Sandhurst, Maryborough, and other localities, failed to yield gold.

"It is a point of great interest to determine the constituents of Victorian mine-waters, as tending to throw light on such questions as these introduced, and as to whether the large nuggets found in the drifts have been built up by continuous aggregation of precipitate from mine-water. Microscopic examination of thin sections of such nuggets should be obtained. It should also be ascertained if they enclosed foreign material. That the waters percolating our drifts have, in many instances, a strong solvent action on some metals and metallic oxides, we have constant evidence in seeing the blue slates of the hill-slopes, where covered with drift in the valleys, converted into white pipe-clay. Was this solvent carbonic acid, or have we at times a stronger acid in operation capable of acting on silver, and so affording a reason for the fact that the alluvial gold of a district usually assays higher than the reef-gold of the same district?

"Sufficient evidence has, I think, been shown, that, up to comparatively recent times, solution and re-deposit of gold and silver have taken place; although the main concentrated deposit occurred with the accumulation of the quartz itself, in the reefs, still much of the 'casing' gold on the walls of reefs may be of subsequent deposit. If, then, it is a logical deduction from what has preceded, that auriferous quartz reefs have derived their minerals from the bounding rocks, and that auriferous quartz reefs may be of all ages, how shall any one assert that drift-gold may not be found in any sedimentary deposit, derived from rocks traversed by auriferous quartz reefs? At the same time, it is likely to be rare that workable gold-deposits will be found in any marine beds derived from pre-existing rocks, unless entirely made up of the débris of such auriferous rocks. The Nova Scotia Carboniferous conglomerates answer this condition, and are worked, according to Hartt, to a profit. In Victoria, no strictly marine sediment has yet been found to contain workable gold-drifts. All our Tertiaries in which marine fossils have been found, also the 'Lower Mesozoic' of Bacchus Marsh, and the Mesozoic carbonaceous of Otway, Cape Patterson, &c., are composed of sediments of various rock formations. We have as yet no evidence to prove that any workable auriferous drift deposit, of Victoria, has been swept by either of the oceans which deposited strata containing marine fossils in other localities. This seems also to be the case in California. Whitney says:—

'The vast Tertiary formations on the flanks of the Sierra Nevada, so important as being the locality of the hydraulic mining operations, are not of marine origin, as has been so often asserted. . . . In the first

place, these deposits are not of marine origin, as is proved by the fact, that, although frequently found to contain impressions of leaves, masses of wood, and imperfect coal, and even whole buried forests, as well as the remains of land animals, and, occasionally, those of fresh water, not a trace of any marine production has ever been found in them.' ”

Wilkinson on Formation of Nuggets.

The next important contribution is Wilkinson's paper on the Theory of the Formation of Gold-nuggets in Drift, written in 1866.

“It has hitherto been a moot question, and one which has elicited no small degree of discussion, respecting the occurrence of larger nuggets of gold in the drifts than have yet been discovered in any quartz reef; and that alluvial gold is generally of a higher standard than that obtained from the reefs.

“Many theories have been introduced to account for these phenomena: among them is one which does not appear to have received that amount of attention it evidently merits. I allude to that advanced by Selwyn, the Government geologist, suggesting the probability of gold existing in solution in the mineral water permeating the Silurian rocks and the gold-drifts; and that this water in its passage through the drifts, became by some unknown means decomposed, influencing the precipitation of the gold, which concreted, so to speak, around the most congenial nuclei presented to it, such as the particles or pieces of reef-gold existing in the drifts, or any other metallic substances for which it had an affinity.

“Daintree, formerly of our Geological Survey, had on one occasion prepared for photographic use a solution of chloride of gold, leaving in it a small piece of metallic gold undissolved. Accidentally some extraneous substance, supposed to be a piece of cork, had fallen into the solution, decomposing it, and causing the gold to precipitate, which decomposed in the metallic state, as in the electro-plating process, around the small piece of undissolved gold, increasing it in size to 2 or 3 times its original dimensions.

“Considering this accidental experiment of Daintree's as in some measure bearing out Selwyn's hypothesis, I was induced to make a few simple experiments.

“Using the most convenient salt of gold, the terchloride, and employing wood as the decomposing agent, in order to imitate as closely as possible the organic matter supposed to decompose the solution circulating through the drifts, I first immersed a piece of cubic iron-pyrites taken from the coal formation of Cape Otway, far distant from any of our gold-rocks, and therefore less likely to contain gold than other pyrites. This specimen (No. 1) was kept in a dilute solution for about 3 weeks,

and is completely covered with a bright film of gold. I afterwards filed off the gold from one side of a cube crystal to show the pyrites itself, and the thickness of the surrounding coating, which is thicker than ordinary note paper. If the conditions had continued favourable for a very lengthened period, this specimen would doubtless have formed the nucleus of a large nugget. Crystals of gold have been found to contain nuclei of brown iron-ore and undecomposed iron-pyrites.

"No. 2 specimen contains iron-pyrites, and was immersed in a solution of about 4 gr. of the chloride of gold to 1 oz. of water; in a short time, however, it was found that in such a strong solution, the pyrites began to decompose; but after diluting to about 2 gr. to 1 oz. of water, this decomposition apparently ceased, and metallic gold deposited wherever a particle of the sulphide existed, alike in crevices as on the surface of the quartz, and also in a remarkable mammillary form. This was in the solution for a week.

"No. 3 contains iron-pyrites and galena, on both of which the gold has deposited, so that I cannot now distinguish one sulphide from another. It remained in a solution of 1 gr. of chloride to 1 oz. of water for 8 days.

"Nos. 4 and 5 are similar specimens to the last mentioned, the same strength of solution being used; but they were only dipped half-way into it, so that the immersed part coated with gold may be compared with the other half on which the pyrites remains unaltered.

"I may here remark that a weak solution produces more perfect results than a strong one; with the latter, the sulphides are partly decomposed, and the gold is covered with a dark-brown powdery film, as observed in some of the above specimens. This film does not prevent the growth of the gold in the solution, and it may easily be rubbed off.

"Nos. 6 to 13.—Iron-, copper-, and arsenical pyrites, antimony, galena, molybdenite, zinc-blende, and wolfram were treated in the above manner with similar results.

"Brown iron-ore and quartz covered with peroxide of iron were also tried in the same way, but the gold was deposited only as a fine metallic powder.

"In the above experiments, a small chip of wood was employed as the decomposing agent. In one instance I used a bit of leather. All through the wood and leather, gold was disseminated in fine particles, and when cut through, the characteristic metallic lustre is brightly reflected.

"The first six of these sulphides were also operated upon simply in the solution without organic matter, but they remained unaltered.

"Iron-pyrites was tried with metallic copper, zinc, and iron as decom-

posing agents; but metallic gold was deposited only as a fine powder, which settled at the bottom of the vessel.

"From these experiments, it would appear that organic matter is the necessary chemical agent to decompose a solution of the chloride of gold, in order to precipitate the gold as a coherent coating around a nucleus presented to it; and that so far as we have yet tried, iron-, copper-, and arsenical pyrites, galena, antimony, molybdenite, blende, wolfram, and metallic gold, constitute especially favourable nuclei to demonstrate this chemical reaction.

"Organic substances, such as fragments of wood, roots of trees, &c., exist abundantly in the gold-drifts. It remains, therefore, a point of great importance to decide whether gold is actually in solution in the meteoric water circulating through our rocks and drifts. I am not aware of direct experiments having been made to solve this question, but that gold will most probably be found, is indicated by analysis made by Daintree.

"I referred to the mammillary form the gold assumes in No. 2 specimen, which appears to be analogous to that presented by the surface of nuggets. Analogy, however, though generally a truthful guide, if relied upon too implicitly in outward semblances, may lead to erroneous conclusions. Nevertheless, the striking similarity in the surface of the artificial production to that of the natural gold is a point worth noticing. For if the form of the latter is the result of abrasion of its surface by the material carried along by the streams that once swept down the courses of our old 'leads,' then our analogy will not hold good. Yet when we have no evidence of the existence of such large nuggets in the reefs, and this theory introduces a means of producing results like those in nature, we are justified, in the absence of such evidence, to attribute these results to analogous causes. Otherwise to what origin shall we ascribe the presence of gold in pyrites that has been formed in wood imbedded in the auriferous drifts, and the fact that sometimes gold encloses a nucleus of brown iron-ore, &c., unless it was deposited from solution?

"That gold may be greatly purified by dissolving and reprecipitating it, is strong evidence in favour of the theory attributing to a similar cause the greater purity or higher standard generally of alluvial than reef gold.

"It would be premature for me to speculate further on the hypothesis of the growth of gold—the formation of nuggets in the drift, on which the above recorded few simple experiments may perhaps throw some light—until the result of more comprehensive and systematic experiments which are now being conducted by Newbery are known. In conclusion, I beg to acknowledge my indebtedness for some points in the foregoing to a Report on the Minerals of Victoria, just completed, by G. H. F. Ulrich, of the Geological Survey."

Newbery on Gold in Drifts.

The experiments just alluded to are those described by Prof. J. Cosmo Newbery, in his paper on the Introduction of Gold to, and the Formation of Nuggets in, the Auriferous Drifts, written in 1868. He says :—

“Before describing my experiments and their results, it may be well for me to give an abstract of the arguments used for and against the denudation theory and in favour of what seems to some a rather ludicrous idea—the growth of nuggets in the drifts.

“Through the kindness of Mr. Ulrich, I have been able to read the latest ideas of the eminent chemical geologist, Prof. Bischoff, from whom I shall freely quote.

“That some portion of the gold found in the drifts has been derived from the quartz reefs at the same time that the reefs themselves were formed, there can be no doubt ; but the absence of large nuggets in the reefs, and the marked difference that exists between much of the drift gold and that from the reefs, tends to make us believe that some portion of it had some other origin, or was transferred from the reefs to the drifts by some means other than denudation. Even if we admit that the large nuggets may have been derived from the reefs by denudation—(for there is a theory that the reefs were much richer in the portions removed to form the drifts, than they are as they now exist)—we must remember that the nuggets consist of nearly the heaviest known matter, offering but a very small surface of attack, when compared with the other materials acted on by the same force and at the same time ; it therefore appears strange that these heavy masses should be found at such great distances from any known reef, as nearly all the large nuggets have been. Another point which attracts attention is, that they are sometimes found in the sand overlying the gravel, which is quite inexplicable, if they ever were in motion with the rest of the drift, which usually has a regular arrangement from top to bottom : first clay, then sand, and fine and coarse gravel.

“These objections to the denudation theory are not easily explained away. And then comes the great fact that gold is contained in the iron-pyrites which is found in the drifts, assuming the form of roots and branches of trees, and also replacing the carbonaceous matter of the other drift-wood. Every sample of this pyrites that has been examined has been found to contain gold : in some instances, in a quantity equal to 40 or more oz. per ton, and this in samples in which no particles could have collected in crevices or cracks.

“This proves that gold did exist in the meteoric waters which deposited the pyrites in Tertiary times.

“Based on these arguments, Selwyn, some years ago, advanced the

hypothesis, 'That nuggets may be formed and that particles of gold may increase in size through the deposition of gold from the meteoric waters percolating the drifts, which water, during the time of our extensive basaltic eruptions, must have been of a thermal and probably of a highly saline character, favourable to their carrying gold in solution.'

"As Ulrich points out in his essay on the Mineralogy of Victoria, this view of the character of the meteoric waters in earlier times receives aid from the fact that on our western gold-fields only, where tremendous basaltic eruptions have taken place, all the large nuggets have been found, while on the eastern and northern fields, where basaltic rocks are wanting, or only of very limited extent, the gold is usually fine, and nuggets of more than 1 oz. in weight are very rare.

"That gold does exist in solution in some saline waters of the present day has been proved by several analyses; and Daintree found gold in solution in water taken from a mine in this colony.

"Further proof of gold having been in solution at a comparatively recent date, I found when examining the pebbles of the Miocene drifts; they are chiefly quartz, and are coated over with manganiferous brown iron-ore, in which I found gold, though I never could detect any in the pebbles when their surfaces were carefully cleaned.

"What the gold salt was, whether a chloride, silicate, or sulphide, we have no means at present of ascertaining. And as it may have been in the same solution that deposited the pyrites, which probably contained its iron in the form of protocarbonate with sulphates, it was not easy at first to imagine any ordinary salt of iron; but this I find can be accomplished with very dilute solutions in the presence of an alkaline carbonate and a large excess of carbonic acid, both of which are common constituents of mineral waters, especially in Victoria. This is true of chloride of gold, and if the sulphide is required in solution, it is only necessary to charge the solution with an excess of sulphuretted hydrogen; in this manner, both sulphides may be retained in the same solution, depositing gradually with the escape of the carbonic acid.

"Prof. Bischoff has suggested the occurrence of sulphide of gold in meteoric waters, and by experiment he found that it was slightly soluble in pure water. Once formed and present in the water, it is, like all other gold salts, easily decomposed. In an experiment I have made, the sulphide of gold was held in a solution by a small quantity of an alkaline bicarbonate. A cube of iron-pyrites and a chip of wood were introduced, and in a few days small irregular grains of metallic gold were deposited on the pyrites.

"What part the organic matter took in the reaction is not clear, but the gold was not deposited without it.

"In Chas. Wilkinson's paper (see pp. 14-16), a series of experiments

are described in which gold was deposited in the metallic form upon a nucleus, from a solution of the chloride, by the reducing agency of organic matter, the nuclei being either gold itself, or iron-, copper-, and arsenical pyrites, galena, zinc-blende, sulphide of antimony, &c. Organic matter has long been known as an agent for precipitating gold in the metallic state from its solutions.

“Rose states that oxalic acid precipitates it in metallic laminæ. This I have failed to produce. When boiled with a solution of chloride, I got purple and red precipitates; but when allowed to remain at the temperature of the air for some hours, a film of gold floated on the surface of the liquid, and the bottom and sides of the vessel were gilded. Tartaric, citric, and other organic acids have much the same effect. With wood, bark, charcoal, and like substances, the reduction is much slower. No carbonic acid is seen rising, and the gold is deposited in the pores of the reducing agent, if the solution is dilute. But it was not known until the experiment of Daintree, and the following ones made by Wilkinson, that this deposit would take place on a nucleus, and be continued as long as gold remained in solution. If this action went on in the drifts, it would account for the greater purity of the gold and for the nucleus of brown iron-ore so often found in nuggets and crystals. Strong solutions of gold immediately began to decompose the pyrites, and interfere with the regular deposition of gold. By a strong solution, I refer to one containing more than 1 gr. of chloride of gold to 1 oz. of water. A weaker solution than this also decomposes the pyrites, but so slowly as not to interfere with the deposit taking place regularly; all the other sulphides are also decomposed. In the experiment in which galena was used as a nucleus, this decomposition was best marked. Somewhat more than a year ago, I placed a cube of galena in a solution of chloride of gold, with free access of air, and put in organic matter: gold was deposited as usual, in a bright metallic film, apparently completely coating the cube. After a few months, the film burst along the edges of the cube, and remained in this state with the cracks open, without any further alteration in size or form being apparent. Upon removing it from the liquid a few days ago, and breaking it open, I found that a large portion of the galena had been decomposed, forming chloride and sulphate of lead, and free sulphur, which were mixed together, encasing a small nucleus of undecomposed sulphate of lead. The formation of these salts had exerted sufficient force to burst open the gold coating, which upon the outside had the mammillary form noticed by Wilkinson, while the inside was rough and irregular, with crystals forcing their way into the lead salts.

“Had this action continued undisturbed, the result would have been a nugget with a nucleus of lead salts, or, if there had been a current to

remove the results of the decomposition, a nugget without a nucleus of foreign matter. If, instead of galena, we had had a piece of pyrites to start with, the decomposition would have gone on in the same way, but the result would have been brown iron-ore in place of lead salts. This decomposition gives a very simple means of accounting for the oxide of iron, so often found in the nuggets and crystals of gold, the latter especially, as shown by the experiments of the late Dr. Becker, by cutting them in halves, and by their established low specific gravity, and their loss in weight suffered in smelting.

"Finding the brown iron-ore of the Miocene drifts contained gold, I was led to suppose that though I could not make gold deposit on it, I might succeed in making them deposit together, which was the case. I arranged a mass of sand, with chips of organic matter in it, in a vessel, and slowly filtered through it a dilute nearly neutral solution of sesquichloride of iron, containing a few drops of chloride of gold, and as it passed through repeated the dose. This continued for some weeks without any appreciable change taking place; but after some months, thin bands of hydrated sesqui-oxide of iron began to form across the mass about the centre, parallel with the surface. As they increased in size they assumed a botryoidal appearance, like the 'ferro-manganese ore' which occurs in the quartz reefs, and in many parts were coated with a bright film of metallic gold. Every further addition to the mixed solution produced another layer of oxide and gold, so that in time it appeared stratified. If the gold had been continued alone after once having started its deposition, the result would have been the same as in the case of the decomposition of pyrites. On the other hand, if the iron solution was in excess after a deposit of gold had been formed, it would have produced what is so often found in the alluvial workings, a nugget coated with iron-ore, commonly known as 'black gold.'

"This mixed solution is one which we should not expect to find in nature, but there is no difficulty in supposing the transfer of gold with iron that would deposit as oxide, even if we need to introduce carbonic acid. If a solution of sesqui-chloride of iron and chloride of gold are heated together, the whole of the gold, in a very finely divided state, with a portion of the iron as sesqui-oxide, is deposited in a brownish-yellow precipitate.

"Though the processes I have described will account for the formation of nuggets, it does not account for the appearance of gold in pyrites. I have examined about 100 samples, in none of which do I find any tendency on the part of the gold to assume the form of a coating, it being usually in irregular grains, and small octahedral crystals, seldom to be detected, even with the aid of the microscope, until nearly all the pyrites has been oxidised and decomposed. In a few exceptional cases

pieces have been found projecting ; but all tends to prove the priority of the deposition of the gold, and that instead of pyrites having formed a nucleus for the gold, the reverse has in the majority of instances been the case.

“ It may also have been the first to deposit in the drift-wood, for in all the experiments by Wilkinson and myself the organic structure became so impregnated with gold that when ignited (so as to burn off the undecomposed organic matter) a golden model remained. Flies, which fell into some of my experiments, and were useful in keeping up a supply of fresh organic matter, became so thoroughly impregnated that in some cases the finest hairs on their backs and legs were to be seen in bright gold after ignition. Conditions such as these (before ignition) would be very favourable to the formation of pyrites, offering to a ferruginous water containing sulphates, a reducing agent, and congenial nuclei for the crystals to form on. Crystalline gold is very easily made, by simply introducing a chip of wood into a solution of chloride of gold, containing 5 or 6 per cent. of the salt. The crystals are first seen on the surface of the liquid as a thin film, which, as it grows heavier, falls to the bottom, where it assumes a moss-like appearance ; if this is examined under the microscope, it will be found to be a network of octahedral crystals, resembling very closely the gold-crystals from pyrites. These crystals have been repeatedly made in a carefully-closed vessel, so that no dust might enter, and falling on the crystals form nuclei for them. With these crystals I sometimes found irregular pieces of gold, some in places showing planes of octahedrons. In these experiments, as in all the others, organic matter is necessary, the action ceasing when it was removed, starting again immediately with a fresh addition.

“ These experiments are based on the assumption that the gold exists in the pyrites in the metallic form, and not as sulphide, as has been supposed to be the case by some. Daintree got gold in solution by digesting some of the pyrites from Clunes in sulphide of ammonium ; but I have always failed to prove the presence of it as negative evidence against it. I have the result of experiments made by digesting the pyrites with an oxidising agent, washing the residue free from impurities, weighing the gold, and comparing the result with that got from a portion of the same sample made by the ordinary fire assay and finding that they agreed.

“ If sulphide of gold had existed in the metallic waters, we might expect in some cases to find it ; but, as before noticed, it is so easily decomposed, that it is not possible for much to have resisted the heat caused by the basaltic eruptions.

“ I have experiments now in progress which contain the sulphides of iron and gold in solution, but up to the present time without any result,

in this direction. Like some of the others I have spoken of, they may require a year or more to accomplish the end wished for.

"Prof. Bischoff suggests silica as the medium for the transmission of gold to the quartz reefs. Gold, as he points out, certainly has a great affinity for silica, always being found in connection with it in mineral veins in the drifts and even in the pyrites, where I have always found silica as grains, and minute nearly perfect hexagonal crystals; the occurrence of which I have always been at a loss to account for.

"The Professor's experiment is a very instructive one. He reports it as follows:—'On adding to a solution of chloride of gold a solution of silicate of potassa, the yellow colour of the former disappears. After $\frac{1}{2}$ an hour the fluid turns blue, and in time a gelatinous dark-blue precipitate appears, which adheres firmly to the vessel. After the lapse of some days, moss-like forms are to be seen on the surface of the precipitate, like an efflorescence; on exposure to sunlight, no reduction takes place, but after the lapse of some months, if the precipitate is allowed to remain undisturbed under water, a decomposition takes place, and in the silicate of gold, appear minute partly microscopical specks of gold.'

"If this is the method by which the gold reached the lodes, as the Professor argues, the origin of the silica may also be that of the gold. The origin of the former we now believe to be the silicates of the rocks, by the decomposition of which by mineral waters the silica is conducted to the lode cracks. In these silicates, we have therefore to look for gold, and it is possible that it is contained in them as silicate. To prove this is almost impossible, for if we even found the gold, it would be in a quantity too small to determine whether it was in combination or not.

"Silicate of gold is extremely insoluble in water, but if we assume that its solubility is in the same ratio to the solubility of silica as the gold of even our richest reefs is to the silica in the reef, we shall find no difficulty in admitting that silicate of gold may exist in solution.

"In several instances, an amethystine colour has been observed, both in the quartz reefs and in the wash-dirt of the drifts. Aplin tells me that he observed it in a lead of wash-dirt near Beccworth. When pieces of the clay so coloured were first broken out, no gold was to be seen; but after exposure to the light and air for a short time, the colour disappeared, and it was seen to be full of very finely divided gold. Ulrich also tells me that this colour and phenomena were observed by Clement, a successful quartz miner, of Maldon, who described having found dark-blue clayey bands in the centre of a quartz reef, some 10 ft. thick, at a depth of about 70 ft. from the surface. The colour in this case, as in that reported by Aplin, disappeared when exposed to air and light, and gold became visible. It is to be regretted that no chemical examination was made, as there was undoubtedly a compound of gold present.

"Be this as it may, there can be no doubt that nearly all the native sulphides contain gold, especially those which also contain silver. I have found it with this metal in every sample of iron-, copper-, and arsenical pyrites, galena, sulphide of antimony and zinc-blende, which I have examined from the rocks of this colony; and Dr. Percy proved its existence in every sample of galena he examined, even though they contained little or no silver.

"Bischoff, in reviewing facts like these, says that it has been repeatedly proved that in the decomposition of ore lodes, the silver takes part in the oxidation processes, and is removed in soluble combinations. If such ores are auriferous, and after such a decomposition the lodes undergo mechanical destruction, the gold will, as it is in a very minute state, be carried off with the results of the decomposition. The argentiferous character of the native gold, and the auriferous character of native silver, show that though one metal passed into a soluble form and the other remained metallic, the separation was not complete.

"In this very minute state, gold may possess properties differing from those which it has when in mass. Iron, for instance, when reduced by hydrogen from the oxide, has such a great affinity for oxygen, that, if dropped through the air at the ordinary temperature, it takes fire, whilst ordinary iron-filings, under similar circumstances, are not affected.

"It is therefore possible that gold, under certain circumstances, may, by the presence of silica in solution, become disposed to combine with oxygen, and then to form with the silica a silicate of gold.

"If further experiments prove that alkaline silicates favour the solubility of silicate of gold, this silica theory will be open to but few objections, and the difficulties to impede our progress in solving this most interesting problem in chemical geology will be greatly diminished, as it will not require the presence of strong chemical agents, which are not to be found either in the rocks or in the meteoric waters percolating through them."

Skey on Nuggets.

The next to attack the subject of the formation of gold-nuggets in drift was W. Skey, Analyst to the Geological Survey of New Zealand, in a paper read in 1872. He remarks:—

"The first theory broached to account for the presence of these nuggets in drifts was that they had been broken off some rich reef and transported by water bodily to the positions in which they are now found by us. At first sight, this appears very plausible; but there are several considerations which, when allowed to have their due weight, rather tend to shake our belief in its competency to explain the case. These considerations have been discussed pretty freely, so I need not

detail them here, but will only state that, briefly put, the chief of them are as follows :—The large size of many of these nuggets as compared with any of the masses of gold yet found in our reefs ; their position in the drifts, lying as they sometimes do in the upper layers ; and their superior fineness of quality as compared with that of any of the reef-gold found in their vicinity.

“Impressed by these facts, Selwyn proposed another theory for explaining the origin of these nuggets, and one which certainly appears to meet the question upon the particular points just cited (already quoted, p. 14.)

“At the time this idea was broached, nothing systematic or thorough had been undertaken towards investigating this matter as to the probable presence of gold in those meteoric or saline waters referred to, and nothing whatever had been accomplished towards showing any likely means by which gold, depositing from such solutions, would be determined upon itself as a continuous coating, and in such quantity as occasionally to form nuggets of the enormous size we find them in such drifts, nor did Selwyn indeed make any suggestion on this matter ; perhaps considering the initiation of such an idea sufficient for his part, he left the support of it to the ingenuity of chemists, to whom in fact such a labour rightfully belonged ; in reality, so little was known in support of this theory at the time of its evolution, that it seemed in the highest degree chimerical. Since then, however, chemical investigations have given us results greatly in favour of this idea. Thus in the first place, as regards the presence of gold in a soluble state in the waters percolating our drifts, it appears that Daintree found gold in pyrites which had obviously replaced the organic structure of a tree occurring in a drift-bed, and Newbery, Analyst to the Geological Survey of Victoria, afterwards obtained the same results upon other pyrites occurring in a similar manner, both results showing that gold must have been ‘presented to the pyrites in a soluble form.’

“Since that time, gold has been by no means unfrequently discovered to be present in certain mineral and mine waters, and indeed Daintree has recently found gold while testing the water of a mine in Victoria.

“Perhaps, though, the most important communication we have relative to this subject is that of E. Sonstadt, ‘On the presence of Gold in Sea-water’ (*Chemical News*, 4th October, 1872). This metal has indeed before this been alleged to exist in sea-water, but these allegations have not been sustained with such evidence and accompanied with such detailed description of processes employed as entitled them to an unreserved belief on our part. Sonstadt’s experiments, on the other hand, are detailed minutely, and his statements are supported by the results of different processes.

"The amount of gold present in the water taken from Ramsey Bay he states to be very minute, 'less than 1 gr. in the ton,' still the fact of its presence at all in such water is exceedingly interesting as showing an escape of gold from the land seaward, and so confirms the correctness of the various allegations I have referred to respecting the auriferous character of certain of our springs and mine-waters.

"Thus in different ways, the first question involved in Selwyn's theory is answered in a most satisfactory manner.

"As to the means by which the gold present in these waters has been reduced therefrom and aggregated in masses, solid, homogeneous, and occasionally of considerable size, we have no lack of substances certain to be present in these drifts, and capable of effecting the reduction of gold and silver from the kind of solutions likely to be present there. In various kinds of organic matter and in sulphate of iron, we have substances which will effect this with facility; but we have no sure evidence as yet to show that either of these substances will aggregate the gold which they reduce, or locate it in a marked manner, or preferentially, upon the gold already reduced.

"That gold will be reduced by these substances is certain, but all our present experience in regard to the deposition of gold by them shows that gold so reduced will be dispersed rather than aggregated, so that it would appear that nuggets of gold could not well be formed in this manner.

"In our mineral sulphurets, however, we have agents which are not only capable of reducing gold and silver from solution, but besides are capable of locating them when so reduced in coherent and bulky masses.

"I may state that their nuclear action upon gold depositing from solution by aid of organic matter was suggested by Wilkinson, while their competency to reduce the gold from solution without addition of organic matter was shown by me; thus the aggregation of the nuggety forms of gold from solution becomes a still more simple matter, only one reagent being necessary, so that there is a greater probability of such depositions obtaining than were a double process necessary.

"Knowing the action of sulphides, the manner or the mode of formation of a portion at least of these nuggets seems apparent. Conceive a stream or river fed by springs rising in a country intersected by auriferous reefs, and consequently in this case carrying gold in solution; the drift of such a country must be to a greater or lesser extent pyritous, so that the debris forming the beds of these streams or rivers will certainly contain nodules of such matters disseminated, or even topping them in actual contact with the flow of water.

"It follows, then, from what has been previously affirmed, that there will be a reduction of gold by these nodules, and that the metal thus

reduced will be firmly attached to them, at first in minute spangles isolated from each other, but afterwards accumulating and connecting in a gradual manner at that point of the pyritous mass most subject to the current, until a continuous film of some size appears ; this being formed, the pyrites and gold are to a certain extent polarised, the film or irregular but connected mass of gold forming the negative, and the pyrites the positive end of a voltaic pair ; and so according as the polarisation is advanced to completion, the further deposition of gold is changed in its manner from an indiscriminate to an orderly and selective deposition concentrated upon the negative or gold plate.

"The deposition of gold being thus controlled, its loss by dispersion or from the crumbling away of the sustaining pyrites is nearly or quite prevented—a conservative effect, which we could scarcely expect to obtain if organic matter were the reducing agent.

"Meanwhile there is a gradual wasting away of the pyrites or positive pole, its sulphur being oxidised to sulphuric acid, and its iron to sesquioxide of iron or hematite, a substance very generally associated with gold-nuggets. According to the original size of the pyritous mass, the protection it receives from the action of oxidising substances other than gold, the strength of the gold-solution, length of exposure to it, and rate of supply (velocity of stream), will be the size of the gold-nugget.

"As to the size of a pyritous mass necessary to produce in this manner a large nugget, it is by no means considerable. A mass of common pyrites (bisulphide of iron) weighing only about 12 lb. is competent for the construction of the famous 'Welcome nugget,' an Australian find, having weight equal to 152 lb. avoirdupois.

"Such masses of pyrites are by no means uncommon in our drifts or the beds of our mountain streams. The general velocity of the current flowing over such pyritous matters would in all probability be such as would prevent the development of any crystalline form in the gold thus deposited, as we know very well that for such development motion is unfavourable. The form most likely to be assumed by these deposits then would be the mammillary, precisely that in which our nuggets as a rule occur.

"Upon this mode of accounting for the presence of large nuggets in our drifts, their occasional great superiority in point of size to any auriferous mass as yet found in our reefs, and their superior fineness to such reef-gold, admits of easy explanation.

"Firstly, as regards their comparative size, if we only admit that reef-gold is also deposited by pyrites, and if we assume that the strength of the gold-solutions forming these varieties of gold respectively was not greatly different, it is only reasonable to suppose that the gold masses formed in this manner in drift would attain the greatest dimensions, for

in the first place this gold in depositing would certainly aggregate more, as the pyrites in the drifts or river-beds would be less continuous and more sparsely distributed than that in reefs.

“Further, the supply of gold to pyrites lying in these drifts or river-beds (and so exposed to rapidly changing waters) would be far more copious than to pyrites cooped up in a rocky fissure, and so in contact only with water stagnant or nearly so.

“And secondly, as regards the generally superior quality of these nuggets to gold found in the reefs, it will, I think, appear from the following considerations that such a difference in favour of drift gold is to be expected.

“I have previously shown that silver is deposited with greatest rapidity and certainty upon pyrites from solutions which are alkaline from presence of the fixed alkalies or alkaline earths, and that as such solutions are passed from this condition to an acid one, the silver present in them is retained in solution; any gold, however, that may be mixed with such silver is deposited upon this reducing agent, no matter which of these conditions the solvent is in.

“Now this alkaline condition is precisely that in which, as far as we can ascertain, our lodes or rocks must have been at the time of the deposition of the gold and silver now found in them, and this alkalinity would especially manifest itself in those reefs which traverse rocks of a basic nature, such as diorites or serpentines: hence, by the way, the large proportion of silver alloying the gold found in these reefs, as compared with that alloying the gold found in the lodes of our schists or older formations.

“But though the waters percolating our reefs must be to a more or less extent of an alkaline nature, the drainage waters issuing from them will lose a portion of this alkalinity as they are exposed to the air, or to the products of decomposing organic matters, from having absorbed a quantity of carbonic or other acids (sulphuric, humic, &c.), thus in some measure, according to the distance such waters have travelled from their springs, will their condition be changed until their alkalinity may give way to neutrality, or even acidity, either of which conditions is, as I have stated, unfavourable to the liberal deposition of silver along with gold from such waters. Hence it is apparent that from the instant the waters percolating rocks or lodes leave them to form springs, &c., they are continually passing from a favourable condition to one eminently unfavourable for the deposition upon pyrites of what silver they may contain. Consequently the deposition of gold from solution being as we know unaffected, or but slightly so (comparatively), by the condition of the solvent, the great purity of gold deposited from these surface waters is explained.

"The above explanation of the greater purity of our alluvial or drift gold over gold found in the reef is, I think, much more plausible than that which attributes this difference to the interaction of solutions of gold upon the auriferous masses transported from the reef, whereby the silver of these masses is replaced by gold and so removed, leaving the mass correspondingly richer in gold. That this process can be continued until our largest auriferous masses can be thus affected throughout appears to me impossible when we consider the imperviousness of such metallic masses to liquids, and how nearly the atomic volumes of gold and silver approximate. That a superficial change, however, in this direction may occur is by no means improbable, but such would escape detection unless it were especially sought for. Thus the hypothesis advanced by Selwyn as to the manner in which the nuggets of our drifts may have been formed receives support upon all those points which appear of any importance.

"That nuggets of some size may, however, be in a few instances transported bodily from these matrices into the drifts or watercourses is by no means improbable, but in this case they would, I think, partake of the usual quality of the reef-gold of the country about, and so would be inferior in this respect to gold formed in the manner above described.

"Whatever may be the origin, however, of any particular nugget, or of nuggets generally, when we consider the auriferous nature of many mine-waters, also that of sea-water, together with the decomposing and aggregating action of metallic sulphurets upon the gold of these waters, we cannot avoid the conclusion that gold is now being deposited and aggregated in many of our drifts, and that such depositions have been going on from remotest times.

"In conclusion, the questions as to the source of the gold of our nuggets, the nature of the agencies by which it is dissolved, and the precise chemical state in which it exists in our auriferous waters, are subjects which it is not incumbent upon me to discuss here. I will, however, take leave to make a few observations upon them now.

"As regards their source I think this is rather in gold as disseminated in certain of our slates, sandstone, or schist rocks, than in that of our reefs.

"In reference to the nature of the solvent, I have shown that sulphuretted hydrogen attacks gold at ordinary temperatures, forming a sulphide of the metal, and we know that all the sulphides of this metal we have to this time formed are soluble in alkaline sulphides; therefore, as both these agents are generally present in waters situated at some depth in our rocks, we may very reasonably suppose that a portion, if not all, of our gold has been brought into solution by these agents.

"The state to which such auriferous solutions might pass when

exposed to air and carbonic acid is not easy to determine, but of this we may be certain, that it could not well be one unfavourable to the exercise of the reducing properties of metallic sulphurets upon the gold compound present in them."

Newbery on Vein Gold.

In more recent laboratory reports, Cosmo Newbery returns to the subject of the deposition of the gold in veins. He says:—

"The main practical interest in this subject is the attempt to discover some of the natural laws by which the deposits of gold and auriferous minerals in the quartz lodes and auriferous strata have been governed, and by so doing to aid the mining interests of the country. Most geological observers are agreed that our auriferous deposits are of aqueous origin; in fact that is the only way of reconciling the various phenomena. And observations have proved that gold is deposited in and with recent mineral formations. The first question is—What sort of a solution is necessary to carry the gold and associated minerals, and have we in the natural waters of the mines this necessary solvent? This fluid must have the power of holding a number of minerals in solution all the same time, in such a manner that they may be all deposited together, or at any rate holding certain elements or compounds which, by reacting on minerals already deposited, will produce the minerals as we find them.

"II. Müller, in his description of the ore deposits of Freiberg, notes that the deposits of ore vary with the rock through which the lode passes, and Von Cotta has adopted the term 'ore-carriers' for those rocks exerting a favourable influence on the deposit of ores. He remarks that, although the lodes are not always rich while passing through the rocks conducive to the deposit of ore, and indeed are very frequently barren, still when the lodes do contain ore it is only when in these rocks. This all points to these rocks either supplying the waters with their mineral contents, or exerting some influence to make them deposit what they held.

"Ulrich has on several occasions drawn attention to the necessity for the study of the rocks through which the quartz reefs pass, and the probability of the 'shoots' of gold being in some way connected with the nature of the wall-rock of the vein where the shoot occurs. He says that, although Victorian reefs on casual observation appear to be layers running with the strike and in the bedding of the Silurian rocks, closer examination shows that this is not the case—they are a variety of layer lodes. The greater number certainly lie within the line of strike, but underlie at varying angles through the bedding of the rock, a feature

which results in the 'reef' passing more or less gradually from one bed to another. The greater the angle of difference, and the more regular the dip of the reef in strike, the quicker this passage will be. The points to be ascertained are—Do one or more particular beds of rock enclose the shoots of gold? and if so, does the rock in any way differ from that enclosing the barren parts of the reef? The physical character of the rock, as well as its mineral composition, may play an important part. Ulrich mentions that the richness of the reefs in the Upper Silurian rocks is often increased when they pass into or are in the vicinity of a certain class of rocks which have a mineral composition differing from the ordinary shales, &c. ; while in the Lower Silurian, the differences in the rock are apparently only physical. In some instances, it is found that, where the rock becomes soft, the quartz reef widens, and is richer in gold than where the rock is hard and the reef narrow. This is reversed in quite as many, if not more cases, the reef being poor where it is wide in soft rock, and rich where it is narrow in hard rock. When the latter occurs, it is usually assumed that a simple concentration has taken place ; but there are so many instances of the richness increasing with the width of the reef that some other reason must be sought for.

"Where the mineral components of the rock change, as with the occurrence of dykes and masses of intrusive rock in the Upper Silurian of Wood's Point, Crossover Creek, Walhalla, Alexandra, &c., a notable increase in the richness occurs in the quartz as it passes from the slates towards or into the dyke. By the examination of carefully prepared microscopic sections, Ulrich shows that these rocks having a good influence are of one special variety—they are hornblendic, true diorites ; while all the rock masses and dykes, which are augitic diabase, have hitherto been proved to be non-auriferous. Until Ulrich's observations, all these rocks were classed and mapped as diorites. It is very difficult to distinguish one from the other except in microscopic sections. So far as the examination has gone, this rule of the diorite favourably influencing quartz veins, and diabase having no such influence, holds good not only for Victoria, but also for Tasmania and Queensland.

"Where decomposition or changes of character, either chemical or physical, have been caused by aqueous action, there is a difficulty in separating those which may have taken place at the period of the formation of the quartz veins from those of more recent date. In some instances the actions in the past may have differed materially from recent actions ; but all the evidence we have must incline us to the belief that in the majority of cases the actions were similar."

Again—

"Some auriferous specimens examined lately have afforded further evidence as to the probable deposition, at very recent periods, of at any

rate part of the gold of the alluvial and quartz mines, and of a deposition now going on in many mines.

"Considerable difficulty is experienced in conducting the experiments on this subject, as extreme care is requisite to exclude all possible chance of the presence of finely-divided gold which has been held in suspension in mine-water, either by itself or with pyrites, quartz, and earthy matter. A sample of mud deposited from the water of a mine on the Hustler's line of reef, Sandhurst, was examined by careful washing, and it was found that the heavier portion consisted of auriferous pyrites and free gold. The particles of the latter were of sufficient size to be recognised by means of the microscope. They were irregular grains apparently flattened, having been in all probability reduced to their present condition by the wear and tear of mining operations, or perhaps in part derived from the clayey casings of the quartz lode. This mud was not pumped out of the mine with the water, but had been deposited in the workings. Other instances of auriferous mud carried many miles by running water are not wanting. Samples collected from the silt in the creek, several miles below Clunes, gave an appreciable amount of gold to an ordinary assay of 1000 gr., and were also found to contain iron-pyrites. These and other similar determinations have made it necessary to discard a certain portion of the evidence which had been collected to prove that the mine-waters of the present day carry gold in solution, and deposit it whenever the circumstances are favourable, either in the quartz veins or alluvial drifts.

"The following determinations (qualitative) have been made :—1st—Incrustations from steam-boilers using mine-water at Maryborough. The residuc, after removal of all soluble matter, showed no gold with the microscope, and but few grains of silica. It was a fine powder of grey-brown colour. Fused with pure litharge and assayed, it gave an easily recognisable button of gold. The incrustation consisted of chloride of sodium, sulphate of magnesia, and sulphate of lime. 2nd—Mine timbers collected from the following mines :—

No. 1.—Spanhake Company, Ballarat.	No. 7.—	} Hustler's Reef, Sandhurst.
No. 2.—Black Hill " "	No. 8.—	
No. 3.—Gravel Pits, "	No. 9.—Thanet Mine, "	
No. 4.—Energetic " Sandhurst.	No. 10.—Koch's Pioneer, "	
No. 5.—Koch's Pioneer, "	No. 11.—Bute Mine, Scarsdale.	
No. 6.—Great Extended Hustler's, "	No. 12.—Reliance, "	
	No. 13.—Ellesmere claim, Scarsdale.	

"The samples of timber were for the most part quite sound ; one or two were slightly decomposed for somewhat less than an inch from the surface. Though the whole of them were traversed in all directions by cracks, only to be detected by thoroughly drying them, these cracks were in all instances found to contain ferruginous clayey matter, and in several

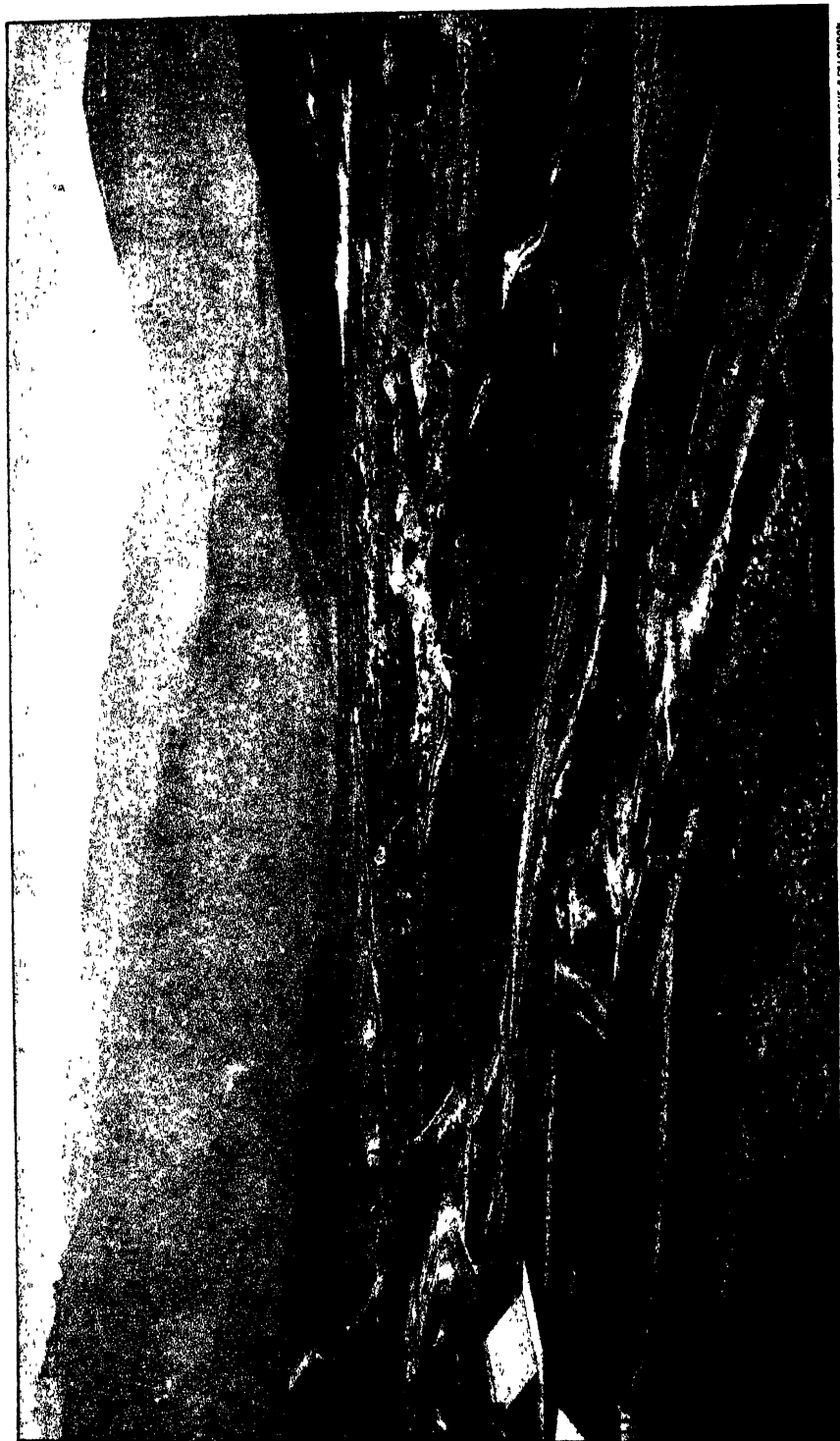
cases iron-pyrites. The soundest samples were taken and burnt in a clean new muffle, first having split the timber into small pieces, and cut away as carefully as possible all the outer portion, the clayey cracks referred to, and any parts that were soft or decayed, which might have taken up by absorption any auriferous mud while the timbers were in the mine. The ash obtained was of a red colour, owing to the presence of oxide of iron. It was washed with water and weak hydrochloric acid, and then examined microscopically. No gold was detected, but it might have been easily hidden by the earthy insoluble matter of the ash. The whole was then fused with a weighed quantity of pure litharge, and cupelled, and gold was detected readily. The button was dissolved, and confirmatory reactions obtained from the solution.

"These experiments were repeated with similar results. In the residues examined by the microscope, were seen a number of angular grains of silica, and as these rendered it possible that some crack or fissure which had absorbed muddy water had been overlooked, some thin chips were cut from the wood where it was perfectly solid; these were burnt, and a similar red ash with the grains of silica was obtained.

"Another quantity of chips was boiled in water, and yielded sulphates and chlorides of the alkalies and magnesia; these were completely removed by successive waters, and the wood oxidised, so as to form sulphuric acid from any pyrites that might be present. This acid and iron were then detected in the solution. These tests prove that gold had found its way into the dense unchanged wood of the mine timbers, and that it must have got there in solution; for it is impossible to suppose that grains of gold and pyrites can have been drawn into the wood by capillary action. The grains of quartz were angular and transparent; and I can only explain their presence as has been done in the case of other wood in which they are found, and which has not been in mines, by supposing them to have been enclosed during the growth of the tree.

"By discarding all the soft portions of the wood and the cracks from these tests, I have omitted all the parts where chemical action would be most active and the best results obtained: 8 lb. of specimen No. 10 gave 0.002 gr. of parted gold; an average of specimen No. 13, weighing $8\frac{1}{2}$ lb., 0.002 gr. of pure gold: 4 lb. of the external soft wood gave 0.01 gr. of gold. This shows a much larger amount of gold than the internal portions; and as the pyrites obtained is crystalline (not loose broken fragments), showing a surface resembling what is termed 'secondary' pyrites, and is firmly fixed on and in the woody matter, I believe it is due to chemical deposition, and not accidentally absorbed.

"I cannot at present say whether the gold is in the pyrites or as free gold; but as I have been unable to detect any microscopically, I think it is with the pyrites.



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"Attempts have been made to determine the actual presence of gold in mine-water by depositing it on plates of copper connected with a galvanic apparatus; but, though gold was found in the crust which coated the plates, the trial could not be relied on, as no test had been made of the copper before the experiment was begun. These experiments will be repeated, with a specially devised apparatus, so that not only shall the copper be pure, but the flow of water made to pass through a filter in order to remove any suspended matter.

"It is also proposed to make some trials similar to those by which Sonstadt proved the presence of gold in sea-water. The evidence of its presence and the continued deposition may be classified under the following heads:—

"1. Gold found with pyrites in the wood of the deep leads, and in the charcoal enclosed in the lower parts of the basalt flow covering the leads, first noted, I believe, by R. Daintree.

"2. Gold found in the trunks of trees converted into pyrites in the same leads, and in the pyrites cementing together the quartz pebbles of the drift, as well as the pyrites forming stalactitic masses on the lower side of the basalt over the lead.

"3. The incrustations from steam-boilers fed with mine-water.

"4. The gold found in mine-water by Daintree.

"5. Gold and pyrites in mine timber.

"6. Gold in the secondary pyrites coating the joints and filling fissures in the quartz of the lodes.

"Some persons who have reviewed the experiments made by Daintree and Wilkinson, have been inclined to doubt the part played by organic matter; and William Skey, of the Geological Survey of New Zealand, states that the action of organic matter in the solutions of gold is simply to cause a general reduction and distribution of the gold, causing it to deposit on the bottom of the vessel as a powder, but not in any way aiding in the formation of nuggets or crystals. I have tried many experiments with organic matter in solution of gold, and though some have been as he describes, many have been otherwise; and I still believe that organic matter has played an important part. In examining the organic matter which I have had in the solutions, I find that it becomes impregnated with gold, so much so that when cut it has a metallic lustre, and may be burnished, and yet little gold was deposited as a crust on the vessel or as a powder. In solutions where no nucleus was placed for the gold to deposit on, films were formed, which floated on the surface, gradually increased in size and thickness, and then sank; on examining these, they were found to be masses of small crystals. Unfortunately, I did not see Skey's interesting papers till long after they were published, and have only lately returned to these experiments. The crystalline

gold has been replaced in the solutions with and without organic matter. Gold plates, having had the surface roughened by the action of acid, have also been placed in similar solutions of chloride of gold, and the results will be duly recorded.

"Like Skey, I have not found that hammered discs of gold increase in size, but I have little doubt of the others with a rough or natural surface doing so. Daintree's discovery, upon which all these experiments have been based, the observations of which Skey somewhat severely criticises, was not made with a hammered disc of gold, but with a rough fragment. Ulrich, who was present when Daintree discovered the enlarged piece of gold, related the circumstances to me thus. The gold originally placed in the bottle was a small fragment which had remained undissolved after making some chloride, and the bottle was closed with a cork ; when again observed, the solution was colourless, and the fragment of gold of such a size that it could not be removed from the bottle, through the narrow neck. Finding that a part of the cork was floating in the bottle, it was suggested that the organic matter had caused the gold to deposit, and that the organic matter of the auriferous leads might play an important part in the formation of nuggets. Wilkinson then successfully repeated the experiment.

"Upon examining numerous samples of auriferous Tertiary or recent pyrites I find that the gold is distributed throughout the mass, and that there is really no more evidence of the pyrites having acted as a nucleus for the gold than there is for the reverse, i. e. the gold for the pyrites. Further, we have, as far as I know, no evidence in nature of gold coating pyrites ; it occurs in pyrites, or attached to pyrites. In fact they are found imbedded in each other, but with no further relation than that they were deposited together. Now we know that, under ordinary circumstances and from laboratory solutions, gold is deposited on pyrites, and, as Skey has pointed out, the pyrites proportionately decomposed ; there must have been other conditions and affinities which prevented our laboratory reactions taking place, for we do not find gilded pyrites. Brown iron-ore, quartz, and iron-pyrites have been found separately in the centre of gold-crystals ; may not all these minerals have acted simply as inert nuclei ? Skeleton crystals of gold in the form of the rhombic dodecahedron have also been observed over the pentagonal dodecahedron of pyrites, showing that the process here involved is actual crystallisation, and not plating or coating, which would have merely covered the faces of the pentagonal dodecahedron."

Whitney on Metalliferous Ores.

Prof. Whitney (1880) states it as his confirmed opinion, after many years of careful examination of important mining regions in various countries, that the occurrence of metalliferous ores is rather a surface phenomenon than a deep-seated one. "The conditions favourable to the formation of veins and vein-like or segregated masses are more likely to have existed near the surface than deep down below it. Such conditions would be—in part at least—diminution of temperature, relief from pressure, and, in the case of true veins, the existence of fissures. The history of mining operations shows beyond dispute that bodies of ore occurring in the segregated form are, on the whole, not to be depended on for persistence. And even true fissure-veins must eventually give out in depth, if for no other reason than the change in the character of the enclosing rock brought about by intense heat. Neither would fissures be likely to continue to exist, nor the materials filling them to retain a distinct form, where the temperature was above the melting-point. Thus, whatever theories we may adopt for the formation of mineral veins, we are led to the conclusion that they must, as a general rule, be better developed near the surface than at great depths. The fact that in some important mining regions the very upper portion of the veins has been oxidised, and then dissolved away by water, is very easy of comprehension, and not in conflict with what has been stated above.

"The fact is not to be ignored that the two by far most important and productive gold-mining districts of the world are regions of former intense volcanic activity. Australia and California exhibit the same phenomenon of rich auriferous detritus buried beneath masses of lava. Although the eruptive rocks, in these cases, are not the direct repositories of the precious metal, there would seem to be strong reasons for believing that there is a genetic connection between the volcanic activity and the enrichment of the adjacent strata.

"The ranges of the Andes and of the North American Cordilleras certainly surpass all other regions in the world in metallic wealth; for hundreds of years they have been supplying the world with a large part of its silver, and no inconsiderable proportion of its gold. The richness of the Peruvian mines long since became proverbial, and the word 'bonanza' is now familiar to all from its association with the mines on the Comstock lode and others of the Far West. These regions have been the seat of the most intense volcanic activity during Tertiary times, and in all probability in previous geological epochs. It is hardly possible therefore, in view of all the facts, not to admit that there is likely to be some genetic connection between the manifestations of

igneous forces and the impregnation of the rocks with the precious metals and metalliferous ores.

"Quite a number of theories relating to the geological epochs at which the various metals have made their appearance, or been introduced into the formations where we now find them, have been put forth by chemists and geologists. Gold has been specially favoured in this respect. No theoretical views regarding this metal have been so widely promulgated and generally accepted as those of Murchison. According to this eminent authority, gold in paying quantities is exclusively confined to the Palæozoic rocks, into which, however, it was introduced at a very late geological epoch. It was also a favourite dictum of this geologist that auriferous quartz veins are a superficial phenomenon, and that mines of this metal would not hold in depth as persistently as those of other metals. The discoveries of the California Survey in regard to the age of the gold-bearing formations of that State have entirely refuted the (until 1864) generally accepted theory of the exclusively Palæozoic age of rocks of this kind, although this fact was not admitted by Murchison in his latest publications.

"The great depth to which some gold-mines have been wrought, with profit, both in California and Australia, in connection with many other facts observed in various parts of the world, justifies us in asserting that auriferous quartz veins are as persistent, on the average, as those worked for the other metals. That the impregnation of the rock with gold took place at a comparatively recent geological epoch, at least in certain prominent and important mining regions, cannot be denied. These are regions of former intense volcanic activity, and the period to which that belongs is unquestionably Tertiary. In regard to mining districts where gold and other metals and metalliferous ores have been found in considerable quantity, and where there have been no striking manifestations of volcanism, accompanied by ejections of lava, as, for instance, along the greater portion of the Appalachian Chain, and especially on its eastern border, no definite statement can be made in regard to the geological period of the metalliferous impregnation. It would appear, however, that the evidence is, on the whole, in favour of this having taken place at the time of, or shortly after, the upheaval of the ranges themselves. To prove that the rocks of such ranges as the Appalachian and Scandinavian, which are surrounded by entirely unaltered Cretaceous and Tertiary strata, have been the scene of extensive chemical reactions during those later periods, would be a difficult task. Under any circumstances, there is no basis for Murchison's idea that gold was—to use his own words—'the last formed of the metals'; for the impregnation of the quartz veins, or rather its segregation, at the same time with the quartz, into veins or vein-like masses, was merely a collecting together of particles

previously existing in the rock, and not by any means a new creation of them.

"It does appear as if there was some truth in the idea that the finding of large pieces of gold in the gravel is not justified by what we see of the occurrence of the metal in the quartz. It is certain, at all events, that the form of the ordinary nugget is something different from that which is offered by the gold as originally deposited.

"The larger part of the gold contained in the quartz exists in the form of particles invisible to the naked eye; and there are many mines which are producing largely and paying handsomely, where free gold can hardly ever be seen at all in the rock going to the stamps. Indeed, there is a general belief among the miners that, 'specimen mines'—or those where the free gold is segregated from the quartz so as to form handsome specimens—are not likely to be persistent.

"Where the gold is visible to the eye in the quartz, the predominating form which it exhibits is that which is best expressed by the term 'scaly,'—a word used by the placer and hydraulic miners in describing the small, rounded, flattened pieces with which they so frequently meet.

"It seems to be the fact that the gold in the quartz never has the proper 'nuggety' character. The metal does not occur as deposited from solution, in solid, smooth, and rounded masses, but in scaly, foliated, filamentary, arborescent, or crystalline forms. The question arises, then, how has this change been brought about? And connected with this is the inquiry whether there is really ground for believing that pieces of gold, after being separated from their original matrix, do increase in size, either by chemical or mechanical causes.

"The finding of large nuggets in the hydraulic mines of California seems, so far as ascertained by the investigations of the Geological Survey, to be of very rare occurrence. There have been, however, occasional statements, in books and newspapers, of such lucky finds in the ordinary placer mines. Most of these, of course, took place many years ago. The largest nugget of which the writer has ever heard is one said to have been found at Vallecito, in 1852, which weighed 25 lb. It is stated by W. Birkmyre, in his list of great nuggets found in various parts of the world, that there is in the collection of the Bank of England a nugget found in Carson Creek, in 1850, and weighing 18 lb. 3 oz. The above-mentioned find at Vallecito is given on the same authority. Nothing is known of its form or of the character of its surface. Such finds seem to be much more common in Australia than they ever were in California, judging from the lists which have been given in the official publications of the Geological Survey of Victoria.

"Large masses of gold have occasionally been found in the quartz

and in the bed-rock in California. The occurrence of such at Spanish Dry Diggings has been mentioned. Carson Hill is another locality from which similar facts have been reported. As far as the results of his own investigations in California are concerned, the writer is not able to find sufficient evidence to support the opinion that the large size of the nuggets in the gravel presents difficulties requiring the aid of chemistry for their solution. If it be true, as the writer believes, that quartz veins as well as all others, as a general rule, have been richer near the surface than they are at great depths, then the occasional finding of large nuggets in the gravel would not be a matter of surprise. Heavy masses of gold are found, even now, in some of the quartz veins, and somewhat heavier ones may have existed nearer the surface.

"With regard to the manner in which the gold in the quartz loses its characteristic forms, so as to become transformed into the smooth rounded masses occasionally found in the placer mines, there seems to be no theoretical difficulty. In the first place, however, it may be stated that by no means all the nuggets have this character. Many of them exhibit more of their original character than would be expected to be found remaining after ages of pounding between the boulders of the gravel. This is particularly true of specimens collected by Professor Pettec from the hydraulic mines during his last year's investigations, and which have been carefully examined by Wadsworth and the present writer. The same fact has also been stated by Ulrich—who appears to be a close observer—in regard to the Australian nuggets.

"There seems to be no doubt that a 'scraggy'—to use a common miner's term—piece of gold can be transformed into a rounded smooth nugget by a sufficient amount of the right kind of rubbing and hammering, which must have taken place as these great piles of detritus were being shifted from place to place by currents of water. Some of the specimens collected exhibit in the most interesting and convincing manner the transitional form between the rough crystalline form and the smooth rounded one. One in particular, from an unknown locality, purchased by the writer in a shop at San Francisco, has one side almost perfectly smooth, and rounded edges turned over upon the back, which itself is covered with crystalline branchings, still retaining a large part of their original delicacy. It is evident, in this case, that the specimen has been protected on one side, while the other has been subjected to abrasion and pounding, the result being a nugget presenting at the same time and in most remarkable perfection, the characteristic forms of quartz-gold and placer-gold.

"That the masses of gold, when they have been released from the quartz veins and have begun to be rolled about in the gravel, could by any possibility be so situated as to become subjected to any chemical

influences by which their mass could be enlarged, seems—to the writer, at least—highly improbable. That occasionally pieces of the metal may be united by pressure or by hammering between the gravel boulders, and that thus a larger mass may be formed by the union of two or more smaller ones, through purely mechanical agencies, seems not impossible ; and some observations of Wadsworth appear to corroborate this view.”

Egleston on Nuggets and Placers.

In 1881, Prof. T. Egleston, of the New York School of Mines, published his views of the formation of gold nuggets and placer deposits. His paper may be summarised thus :—

“The origin of gold, both in placer deposits and in veins, and especially the origin of nuggets, has been the subject of repeated discussions and investigations. In 1874, I made some examinations of the hydraulic mines of California, and was very much struck with the distribution of the gold throughout these deep placers, which were almost invariably poor on the surface, while gradually growing richer towards the bed-rock. The constant presence of fossil wood, and the large quantity of organic matter contained even low down in these beds, was also remarkable. Not being satisfied with the various theories advanced to account for the formation of these deposits, I began an investigation early in 1879 on the conditions of the solubility of gold and the causes of the loss in working gold-ore in a large way. The researches which I have undertaken show that gold must be considered a soluble rather than an insoluble metal, and that the conditions of solution are such as will be found anywhere where gold is likely to occur, and the solution may take place even under the ordinary circumstances of surface drainage, and may be going on freely even where the presence of gold has never been suspected, and that there are causes enough in nature to produce the solution of the gold in sufficient quantities to account for all the phenomena of both the vein and placer formations.

“The general theory with regard to the formation of these placer deposits and nuggets has been that they were the result of the destruction of pre-existing vein-matter, which does not accord with the facts as shown in the deep placer deposits. The gold in such case would be distributed in layers of unequal richness throughout the bed, the richness depending upon the amount of deposition taking place at any one time, and would not occur in increasing richness from the top to the bottom. Further, every particle of gold, of whatever size, would have a rounded form, resulting from its abrasion against the harder rocks, which is not the case, the small as well as the large grains being of very irregular shape. It must also be borne in mind that most of the veins from which the gold

is supposed to have come had a gangue of quartz. The gold is much softer than the rock; the quantity of precious metal contained in the vein would also be very much less than the rock, so that in the destruction of the formations there would be a very small amount of gold being abraded and ground in a very large quantity of rock. It is therefore likely that the coarse particles of gold, which is so much softer, would be comminuted at least as fine as the rock, and the smaller ones much finer than the rock, so that the difference in density would hardly tend to make a concentration by any subsequent action of wind or water, since the small particles of gold would tend to float away and thus prevent the concentration. Where the large particles are not in sufficient quantity to make an extended natural concentration possible, and where the deposition of the sediment of the rivers is taking place, the result would be a very small quantity of almost impalpably fine gold distributed uniformly in a very large amount of comminuted rock, or a production of clays resembling that used to make brick around Philadelphia, which contains very small amounts of gold uniformly distributed through it. The structure, too, of each one of these particles would be the same as that of the rock with which it was abraded, and would be uniform. It is, however, well known that the grains of gold found in the placers are not uniform; some of them are flattened with rounded edges, others rounded, and most are mammillary, all of which forms are not probable, and hardly possible, under the conditions suggested. A nugget rounded like a water-worn pebble is a great exception in any of the placer deposits.

“While the theory of vein destruction might in some cases account for the presence of gold in small quantities throughout the sands in grains large enough to admit of concentration, it could never account for the presence of large nuggets, which if they had been transported any distance by water would have lost their mammillary form. Admitting the greater size and force of the ancient rivers, it is impossible to conceive that such large and irregularly shaped nuggets as those from Australia, Siberia, and from this country could ever have been so transported by water as to be entirely relieved of all their gangue, without having themselves assumed much more regular surfaces and a more uniformly cobble-stone shape. On the other hand, slow accumulations from solutions of varying strength and a deposition of unequal rapidity continued for a great length of time, accounts perfectly both for the form and for all the attendant phenomena. It is a fact, moreover, that very large masses such as these nuggets have never been found in veins, and are confined exclusively to placer deposits. The detrital theory accounts still less for the fact that in many of the deposits, especially where the bed-rock is soft and porous, the gold often enters

it to the depth of nearly a foot, and it is frequently the richest part of the deposit.

"In 1867, Wilkinson (see p. 14), of Australia, made a series of researches with reference to the effect of organic material on the deposition of gold. Sonstadt (p. 24) also made a series of researches on the presence of gold in sea-water, and found it to be present in the ratio of about 1 gr. to the ton of water, or about \$1 (4s. 2d.) for every 25 tons of water.

"Up to this time gold had always been considered as a very insoluble substance, because it was insoluble or very nearly so in most mineral acids. Ingenious metallurgical processes based on this insolubility have been invented, and are still in constant use; but it does not follow that because gold is not affected by the ordinary acids it is therefore not soluble in other substances much more likely to be found in nature. The action of organic acids and of the alkalis was left out of view, and also the fact that the solution of infinitesimal quantities may acquire great significance in a geological sense.

"Bischoff (p. 18) found that sulphide of gold is slightly soluble in meteoric waters, and much more soluble in a saturated solution of sulphuretted hydrogen in water. It has also been ascertained that chloride of gold in minute quantity in an alcoholic solution may remain in solution in the presence of proto-salts of iron, and that metallic gold is slightly soluble in solutions of the per-salts of iron. But the theories founded on these discoveries supposed that gold was much less soluble than it really is, and that the solution required peculiar agencies and a set of circumstances not likely to occur everywhere. Its diffusion in sea-water was accounted for by the presence of chlorine, iodine, bromine, and of alkalis, and these conditions were not thought to be of general application in the explanation of the phenomena exhibited in mineral veins.

"Selwyn, the Government geologist of Victoria, proposed a theory of solution (already described on p. 14).

"These researches and theories, however, did not attract very much attention, and the old theory of the destruction of pre-existing veins was still adhered to. It is to be observed, however, that when gold does come from the destruction of veins, the surfaces are rounded and worn smooth, as is shown in the larger boulder of quartz containing gold detached from a vein in Venezuela, which is now in the collection of the School of Mines. This is in entire contradiction to the mammillary structure of the nuggets. If they had been transported far by water, they would have been rounded and water-worn to much more regular surfaces. These worn surfaces would of course have been confined entirely to the outside of the nuggets, any cavities existing in the

interior of the piece would have been in the condition in which they left the vein, and the edges of any crystals found there would have been sharp; while in the nuggets, the mammillary form exists even in the cavities of the interior, and even where crystals or the commencement of crystallisation is observed, the edges of the crystal are very often blunted or rounded, showing both deposition and solution on these edges.

"It is also to be noticed that the analyses of nearly all the samples of gold taken from veins show it to be much less pure than the nuggets found in the placer mines of the same district. If the gold of the placers had come from eroded rocks, it would be of the same composition as that of the veins of the district in which it was found. It is well known that most of the gold nuggets are pure, while the gold of the veins is of a much lower grade, containing considerable quantities of silver and other foreign metals. Thus the Ballarat nuggets are 992.5 fine; the Australian nuggets vary from 960 to 966; those from veins in California from 875 to 885; in Transylvania, 600 with 399 of silver; and in Nevada there are some of 554 of gold and 429 of silver, and others only 333 of gold with 666 of silver.

"It must be remembered also that the violence of the old placer currents was very much greater than that of the ordinary streams of these days. The rivers were not only larger and deeper, but more rapid, and the results of their action would have been an almost complete comminution of the gold by its rubbing against the harder rocks. If this were the whole of the process, and no further action had taken place, the gold would be found in the sands in this comminuted condition exclusively, and few if any of the particles would have escaped the battering and pounding process incident to long exposure to rolling rocks, and the deposits resulting from it would be found on the bed of the stream.

"Gold is, however, also found as nuggets, and in small particles in rocks which have never been disturbed from their original positions, but which have been decomposed to a considerable depth, and it then has the same rounded form, occupying positions which make it evident that it must have been formed *in situ*, and never have undergone any abrasive action. The nugget found in 1828, in Cabarrus County, N.C., which weighed 37 lb., and the one found in the valley of Taschku Targanka, near Miask, in Siberia, which weighed 96 lb., were both found under such circumstances in a decomposed dioritic rock. In some few cases, it has been definitely ascertained that the gold has been dissolved and precipitated in the decomposed rocks, for it has penetrated only just so far as the decomposition has allowed it, the yield in gold ceasing entirely at the point where the rock allowed no further filtration; while in other rocks of a more porous nature in the same district, the gold has

penetrated to a depth not yet ascertained. Such a condition of things is not uncommon in the gold-belts of the Southern States.

"Admitting that heavier masses of gold did exist in the veins disintegrated by the ancient rivers, gravity alone cannot account for the bottom deposits (which are often 300 ft. from the surface) being the richest. It would have required greater agitation of the earth than we have any evidence for believing ever took place to sift the coarse particles even through 50 ft. depth of earth, and there is no indication that these deposits after they were once made were ever disturbed. It is undoubtedly true that in shallow placers, where the bed-rock comes near the surface, the surface-soil is rich ; but it is the invariable rule that in the deep placers the richest deposits are near the bed-rock, and at a great distance from the surface.

"There is a tradition, which is prevalent in all the gold-mines of the South, and in those of some other districts, to the effect that gold grows, so that every few years the tailings of the old mines are re-worked, generally with a profit ; the quantity separated each time, according to the local tradition, being in proportion to the length of time the material has remained undisturbed. As there is no opportunity for the gold in these sands to accumulate by gravity, the people of the region believe that gold grows like a plant.

"It would not, however, be rational to deny a theory so easily explained as the formation of placers by the destruction of vein-matter without having some other to replace it. If the theory of the destruction of pre-existing veins is not tenable, we are bound to examine carefully whether there are causes in nature sufficient to account for solution, and what are the agencies that render the gold soluble. A series of experiments have been made on this subject lasting over many months, both synthetical and analytical, which seem to be of considerable importance in the study of the origin both of placer and vein phenomena. In this investigation, most of the known salts of gold were prepared ; but as the chloride is most easily made, this was made the basis of almost all the solutions. While making the chloride of gold for the solutions, some sponge-gold was placed in a tube and heated in a current of chlorine-gas until the chloride of gold formed was entirely sublimed. It deposited at the upper part of the tube directly over the gold, and, as the tube cooled, on the gold also, in fine transparent crystals $\frac{1}{2}$ in. long. This tube, when cool, was closed while full of chlorine, by replacing the glass tubes by glass rods, and the joints were made tight with paraffin. In 5 months, the crystals were melted into a mass ; and in a year, the whole of the chloride had been transformed into metallic gold, with occasional nodules of chloride through it ; but the whole of it could be readily amalgamated.

"In order to ascertain the effect of different organic substances on salts of gold in solution, 5 portions of 50 c.c. each of a solution containing 0.5 *gram.* of chloride of gold were treated in different ways. The first was covered with 1 c.c. of petroleum. In the second, $\frac{1}{4}$ *gram.* of cork was placed; in the third, $\frac{1}{4}$ *gram.* of peat; in the fourth, $\frac{1}{2}$ *gram.* of leather; in the fifth, $\frac{1}{2}$ *gram.* of leaves. These solutions were put in a dark place, and were left for 3 months before examination. When the solution containing the petroleum was brought to the light, the liquid had lost its colour, and there were suspended in it a number of very fine and long crystals of gold, distributed nearly uniformly from the top to the bottom, and floating almost perpendicularly in the water. They had the appearance of the hexagonal crystals described by Professor Chester. As soon as the liquid was agitated, they fell to the bottom. The solutions containing the cork, leather, and leaves had also been rendered colourless, but the gold had entered into these substances, replacing the organic matter, so that they were pseudomorphed into gold. The solution in which the peat was placed was also colourless; but the gold was precipitated in the form of very small mammillary masses, recalling perfectly the form of nuggets.

"To ascertain the degree of solubility of gold a quantity of pure spongy gold was prepared, and placed in a variety of solutions. Some of these were left exposed to the air; others were sealed at the ordinary temperature and pressure of the air for periods of 6 to 8 months; others were exposed to heat and pressure under varying conditions in an air-bath, arranged in such a way that the temperature could be kept constant for a number of hours at a time. Many of these last tubes burst after the liquid had acquired a tint. Of some of these, the contents were entirely lost; of others, a sufficient quantity of the liquid was left to test for gold.

"Solutions of salt, sulphate of ammonia, chloride of ammonium, chloride and bromide of potassium in sealed tubes, after 8 months, gave no reaction. Heated for 5 hours at temperatures varying from 150° to 200° C. (302° to 392° F.), none of them, except the bromide of potassium, gave any reaction, and that reacted very strongly. In the sealed tubes, the solution of salt, in which a few drops of nitric acid had been placed, reacted for gold; the iodide of potassium gave no immediate reaction, but when evaporated to dryness left a purple residue, soluble in bromine, which reacted for gold. Heated to a temperature of 100° to 170° C. (212° to 338° F.), the iodide of potassium tube gave a reaction for gold not much stronger than the solution before heating.

"A solution of commercial nitrate of ammonia, which contained some chloride of ammonium as an impurity, kept in an open tube at the ordinary temperature and pressure for 4 $\frac{1}{2}$ months, coloured the solution

bright-yellow, and reacted strongly for gold. Two solutions were made, each containing 5 *gram.* of nitrate of ammonia and $\frac{1}{2}$ *gram.* of chloride of ammonium in 200 c.c. of distilled water. One of the solutions was left in an open room and the other put in a dark place, and left for 11 days. At the end of that time, both reacted strongly and with equal intensity for gold.

"Pure sponge-gold was then placed in the following solutions, contained in sealed tubes at the ordinary temperature and pressure for 3 months. Sulphide of ammonium produced no change and no reaction. With sulphide of potassium, a black precipitate was formed, and a strong reaction for gold was given by the liquid. Sulphide of sodium gave a black precipitate and a strong reaction for gold. Cyanide of potassium gave a yellow solution, a brown precipitate and smell of ammonia, and a strong reaction for gold. Chloride of magnesium, after nearly 3 months, gave a gelatinous precipitate, but no gold. Sulphate of soda, after the same length of time, produced no change and no reaction. The sulphate of copper produced no change after $2\frac{1}{2}$ months.

"Spongy gold was then put into solutions of the following substances, and heated for $6\frac{1}{2}$ hours between 145° and 180° C. (293° to 356° F.). Sulphide of ammonium showed no apparent change, but reacted strongly for gold. The solution of sulphide of potassium attacked the glass strongly; it looked greenish, and the liquid reacted for gold; a black precipitate was formed, which was dissolved in bromine, and reacted for gold. The solution of sulphide of sodium acted slightly on the glass, and acquired a greenish tint; a pink film was found on the glass, and a slight precipitate was formed. This film reacted slightly, and the solution very strongly for gold; there was not enough of the precipitate to examine. The solution of chloride of magnesium attacked the glass strongly, from which scales fell, but no gold was dissolved. The solution of sulphate of soda gave a cloudy, flocculent precipitate, but no reaction for gold. Commercial sulphuric acid and solutions of sulphate of potash, iron, and manganese gave white scales, but no reaction for gold. Solutions of sulphate and nitrate of soda gave no change and no reaction. The solution of permanganate of potash produced no reaction. In the solution of cyanide of potassium, the brown precipitate which was formed in the previous experiment dissolved, reducing the gold in the solution so that no gold was found dissolved.

"A mixture of nitrate of silver and sulphuric acid produced no change after 2 months. A mixture of the sulphates of potash, iron, manganese, and commercial sulphuric acid produced no change after 2 months. The permanganate of potash and sulphuric acid gave a black precipitate and coloured the liquid slightly pink, but gave no gold.

"In order to test the effect of organic matter in solution, $\frac{1}{2}$ *gram.* of

chloride of gold was placed in 2 *litres* of Croton water in two large bottles. One of these was left exposed to the sunlight, and from this all the gold was precipitated in less than a week ; the other was placed in a dark room and left there ; at the end of 8 months, a small amount of gold was precipitated. When solid organic matter was placed in the bottle, the precipitation was quite rapid ; and when the bottle was then brought into the sunlight, all the gold was precipitated in about 48 hours.

“ To ascertain the effects of the different soils on weak gold-solutions, $\frac{1}{2}$ *gram.* of chloride of gold was dissolved in 10 *litres* of filtered Croton water, and made to pass continuously over the 3 mixtures given below arranged in glass funnels. The apparatus was so arranged that the liquid would flow drop by drop on the filters :—No. 1 contained 30 *gram.* of quartz sand ; No. 2, 20 *gram.* of sand and 10 of soil ; No. 3, 30 *gram.* of magnetic iron sand and 10 of quartz sand.

“ The filters were left exposed in a room where there was considerable dust arising, and where there was also the smoke from passing trains. In 2 days, most of the gold had been precipitated in the filters, and the water had a greenish look. Then $\frac{1}{2}$ *gram.* of chloride of gold was dissolved in 10 *litres* of distilled water, and filtered in the same way over 30 *gram.* of quartz sand, a mixture of 20 of sand and 10 of soil, and a mixture of 10 of sand and 30 of magnetic sand. These filters were carefully covered so as to prevent any dust settling on them, so that they were protected from all organic matter except such as was contained in them. At the end of 2 months, the clean sand and the mixture of magnetic and clean sand had reduced a small quantity of gold (a little more in the latter than the former) in concretionary shapes, which, owing to the rapidity of the action, were not coherent, but could be crushed with the pressure of the finger. In the mixture with the soil, the whole had been reduced, and was distributed through the sand as an impalpable powder, no indication of any concretionary form being observed.

“ The attempt was then made to dissolve gold in a manner similar to that which was supposed to take place in nature. For this purpose, filters were prepared of 30 *gram.* each of clean quartz sand ; in one of these 1·161 *gram.* of sponge-gold was placed and carefully mixed with the sand ; in each of the other two, $\frac{1}{2}$ *gram.* of very fine gold was mixed. Over the sponge-gold, 10 *litres* of distilled water, containing 30 *gram.* of common salt and 5 *gram.* of nitrate of soda, was made to filter constantly for 2 months ; but no observable change took place. For the second solution, 6 *litres* of Croton water were taken, in which 9 *gram.* of nitrate of ammonia and 1 *gram.* of chloride of ammonium were dissolved. This was made to filter constantly for one month, but no gold was dissolved. For the last experiment, 1 *gram.* of nitrate of

ammonia and 9 *grm.* of chloride of ammonium were used, but no gold was dissolved.

"It was the intention to continue these filtrations for 6 months at a time, and with all the conditions of natural waters, but the difficulty of making the experiments continuous decided the abandonment of them after a number of the other results had been obtained. The failure to dissolve gold in this short time does not prove that there is no action, as the other experiments show. An amount equal to a little less than that in sea-water might easily escape detection. In these experiments there is lacking the certainty that all the conditions necessary to success will be fulfilled. It was found in one of the experiments, made in the early stages of the investigation, that the fine dust circulating in the room was sufficient to precipitate the gold from a dilute solution. All these researches had to be made in a room to which many persons had access, and it is quite possible that the organic matter precipitated the gold in these last experiments as fast as it was dissolved; for in the experiments for the production of the placers the organic matter did deposit the gold in the sand. It is greatly to be regretted that these experiments could not have been made in the complete absence of dust.

"In order to test the effect of organic life in such solutions a plant was watered with a very weak solution of gold, but, as is often the case in such experiments, the plant died of too much watering. In the anxiety to produce the kind of absorption by plants described by Durocher and Malaguti, the experiment was made a failure by too much enterprise. The examination of the ashes of the plant showed a small amount of gold, but most of the gold precipitated was in the soil around the plant, being thrown down there by the organic material in the earth. This experiment indicates the origin of the thin plates of gold which are sometimes found in the grass roots of certain placer countries.

"It will be observed that in almost all the cases where gold was dissolved, chlorine and some nitrogenous substances were found together in the presence of alkaline waters. These same conditions are favourable also to the separation and solution of silica. It has been proven by these experiments that the alkaline sulphides act on gold as well as the substances enumerated above, and it is quite easy to imagine the conditions under which the gold, already in solution in excessively small quantities, coming in contact either with solid or liquid organic matter, may precipitate all the metal.

"In Grass Valley, California, I have known of gold being thrown down in the filter of a Plattner's vat by the organic matter contained in the very impure water used there for the solution of the gold rendered soluble by the action of the chlorine. The filter was full of metallic

gold, and there was no means of ascertaining how much of it had been lost. Several ounces of a brown deposit were taken from it, which was nearly pure gold. This cause of deposition, and of loss in large operations, has, I believe, been entirely overlooked. It is quite easy to explain the presence of gold in alluvial sands by the action of sunlight alone on the waters containing the gold in solution ; and to account for the gold on the bed-rock, by the solutions coming in contact with organic or mineral matter, such as the lignites, fossil woods, or the pyrites, which is everywhere found in deep placer deposits. The waters not being able to pass the bed-rock, remain there in contact with the organic matter until all the gold is precipitated. The same would be true of the decomposing rocks, or of slaty strata coming up to the bottom of the deposit at an angle. The deposition would be rendered much more rapid by any electrical currents that might pass through the strata.

“In all of these phenomena, time, which in the operations of nature is unlimited, is one of the chief factors. In any laboratory experiment, the limit of time must of necessity be short, but there is no such limit in nature. That this solution goes on on a large scale there is every reason to suppose. That it may be connected with vein phenomena the California nugget shows, since in this case both the formation of the quartz and of the nugget are evidently posterior to that of the blue gravel.

“It will be seen from these reactions that many of the conditions favourable to the solution of gold are also favourable for the solution of silica, and that, as Professor Kerr shows, the rocks may be actually decomposed and the gold deposited, forming in this way shallow deposits called veins, in which the gold disappears entirely beyond a few feet.

“Nothing is more likely than that the infiltration of water through rocks undergoing decomposition, of which there are enormous quantities in the gold-region, should take up the alkalis, and, slowly passing over the gold, should dissolve it. The composition of these alkaline salts would depend on the nature of the rock through which the waters passed ; but it is more than likely that they would be mixtures of many of the compounds likely to attack the gold and carry it off in solution, and not alkaline carbonates and sulphides alone, although these would be likely not only to be present, but to be powerful agents in carrying on the work of solution. In some cases, the decay of the rocks is so rapid that the phenomena may, as it were, be caught in the act. The agencies producing the decomposition of the rock, penetrating it beyond the limits of local drainage, and carrying off the soluble parts, leave the debris in a condition easily penetrated by the infiltrating solutions, and ready to receive any deposit which these solutions may from any cause

leave behind them. A source of these deposits in the deep placers of California is the trap which sometimes covers the old river deposits to a depth of 150 ft. In the deep placers, these waters would be capable of holding the gold in solution until they met some decomposing element, such as particles of metallic compounds, native metals, or organic matter, which is always present in large quantities in the deep placers. If a nucleus of metal were present, the gold would be precipitated on it, and if none were present, then the gold would come down as a powder, each grain of which, however small, would serve as a nucleus for future aggregations.

"Admitting the solutions to be even more dilute than the sea-water near the coast of England, yet unlimited time and quantity would evidently produce these effects, re-distributing the gold. Iodine, which is a solvent of gold, is found in many of the plants of the gold-region, and in considerable quantities in sea-water. Sonstadt supposes that gold is kept in solution in the water of the sea by the slow rate at which iodate of calcium is decomposed in the sea-water of the temperate zones, but suggests that where the decomposition of the iodate, whose presence is necessary to keep the small amount of gold in solution, is very rapid, as in hot countries, the liberation of the nascent iodine, and consequent rapid solution of the gold, and subsequent precipitation by organic matter, is quite sufficient to account for the great richness of the gold-deposits of tropical countries.

"It seems by the experiments already cited, to be clearly proved that gold is not only not insoluble, but that in nature it is constantly being dissolved out of the rocks and placers, the waters of filtration dissolving out of the rocks in their passage through them all the materials necessary for the solution of the gold, and carrying it in very dilute solutions until it meets some substance that precipitates it. It seems to be proved that when the action is slow and localised, we have the phenomena of placers with large or small nuggets and irregularly shaped pieces; and when the action is rapid, we find the gold in small particles distributed through the sands. We have reason to suppose that these phenomena are now taking place in such a way as to concentrate the gold by infiltration and precipitation in the tailings of mines which cannot be concentrated by mechanical means. Some of these phenomena can be accounted for by the simple action of sunlight; but others, mostly those of the deep placer deposits, have their cause in the large amount of organic material contained in them. The use of a charcoal filter to precipitate the gold from relatively concentrated solutions, in one of the recently invented metallurgical processes, is a very suggestive idea of the means nature may have used on an immense scale on very attenuated solutions.

"The same conditions which cause the solution of gold in certain cases cause also the solution of silica. This explains the phenomenon of mammillary and apparently water-worn nuggets (like that from Placer county) encased in quartz, while both the gold and the quartz have been formed posterior to the blue gravel. It also explains the presence of 'putty stones,' as the soft pieces of decomposed rock constantly found in placer deposits are called. Many of the causes which produce the precipitation of the gold would also produce the reduction of soluble sulphates to insoluble sulphides, the gold being retained in the mass. This would account for the almost constant presence of gold in pyrites, or the occurrence of some of the copper-ores of Texas in the form of trees, the ore containing both gold and silver; also for the constant presence of gold in the iron-ores of Brazil, the so-called *jacutinga*, and for the presence of trees transformed into iron-ore carrying gold in some of our Western States. In many of the deep placers of California, the heavy cap of basalt is quite sufficient to account for many of the phenomena which occur not only beneath but around it.

"The fact that gold has not as yet been found in potable waters may be simply due to the extreme difficulty attendant on its detection in minute quantity. It is more than likely that many of the geological phenomena on a large scale were produced by very dilute solutions or very slight forces acting for a very long time. How far the electrical currents of the earth may have been a factor in these phenomena it is impossible to surmise. It is, however, more than probable that they were the result, not of one alone, but of all the causes mentioned, and perhaps many others which have as yet escaped our attention. No single agent is so powerful a solvent of gold as chlorine. Very few drainage-waters are free from some compound of it, and no soil is without the nitrogenous materials necessary to set the chlorine free, and therefore capable of attacking the gold and rendering it soluble. The experiments show that a trace of it is quite sufficient to dissolve enough gold to colour a solution so that the eye can detect it after a few weeks' exposure.

"In the nugget of Placer county, it would have been impossible for either the gold or the silica to have got into its position except by solution. The iron of the blue gravel in this case seems to have been the first cause of precipitation, and subsequently the gold itself was an active agent in increasing its own weight. The general absence of crystals, and their rounded edges where they are found, can be easily accounted for by the fact of the rapid action possible in the placers. The readiness of filtration through the easily permeated gravel causes the gold to precipitate so rapidly that there is no time for any but a mammillary deposit; while in vein deposits, the extreme slowness of the

deposition allows the gold to assume a crystalline shape. When we consider that two-thirds of all the gold produced in the world comes from alluvial deposits, it seems difficult to account for its presence in the sands in any other way than by solution."

Newberry on Genesis of Gold.

The last writer to be quoted on this subject is Prof. J. S. Newberry, whose paper on the Genesis and Distribution of Gold was published late in 1881. It runs as follows :—

"Gold occurs in three classes of deposits: 1st, placers; 2nd, segregated quartz veins in metamorphic rocks; 3rd, in fissure-veins, or repositories connected with them, where it is mingled with the ores of iron, lead, tellurium, silver, copper, &c.

"Placer Deposits.

"Nine-tenths, and possibly ninety-nine hundredths, of all the gold which has come into the possession of man, have been obtained from superficial deposits. These are called *placers* by the Spaniards, from whom we have adopted this as we have so many other mining terms.

"The gold here occurs in beds of sand, gravel, and boulders, which have plainly been washed down from higher ground, usually from neighbouring mountain slopes, composed of metamorphic rocks, containing auriferous quartz veins. This relationship is so constant that most persons who have observed or read much of placer-gold deposits have been satisfied with the simple and obvious inference that the gold has been derived from the erosion of higher rocks and the breaking down of quartz veins, the detrital materials having been sorted by water according to their specific gravity. Probably this view would have been universally accepted, the array of facts in its support is so convincing, if it had not happened that men are so constituted that it is impossible for all to think alike on any subject. Among other fruits of this constitutional heterodoxy is the coinage of a new theory to account for the genesis of gold in placers, namely, that the grains and nuggets are formed where found by precipitation from chemical solution.

"The facts by which the advocates of this theory have attempted to sustain it are: 1st, the rarity of nuggets or gold masses of any considerable size in quartz veins; 2nd, the greater purity of the fine gold of placers than that of the neighbouring veins; 3rd, the frosted character of the surface of some gold nuggets; 4th, instances of deposition of gold in organic substances buried in the gold placers; and 5th, the solubility of gold as proved by laboratory experiments.

"In arguing from these facts, it is stated that nuggets of gold of

large size have been not unfrequently met with in all the great placers : such as the Sarah Sands and Welcome nuggets of Australia, which weighed respectively 233 and 184 lb. ; the great nugget from Miask, in the Ural Mountains, which weighs 96 lb. ; and others ranging from 25 to 50 lb., taken from the placers of Australia, New Zealand, and California ; while ' no masses of gold of anything like the size of these have been found in quartz veins.' This latter statement is, however, erroneous ; for the largest mass of gold yet found on this continent was taken from a quartz vein in the Monumental mine, at the Sierra Buttes, 12 miles N. of Downieville, Cal. It weighed $95\frac{1}{2}$ lb. and is said to have been originally larger, weighing 140 lb. This discovery is well authenticated, and is reported in detail in Professor Raymond's Report on Mines and Mining for 1870 (Ex. Doc. No. 207, page 63). This, then, was one of the largest nuggets of gold found in the world, and proves that large nuggets can and do occur in quartz veins. Nor is this rarity of nuggets in quartz veins any proof of the truth of this new theory ; for it is certain that the amount of nugget-gold taken from mineral veins is in fair proportion to the whole gold yield of such veins when compared with the proportions of nugget- and dust-gold in the placers. When it is remembered that from the slopes of the Sierra Nevada, over large areas containing auriferous quartz veins, a sheet of material, perhaps thousands of feet in thickness, has been removed by erosion, it will not be surprising that a considerable number of masses of gold have been taken from the resulting débris ; and when we compare the insignificant excavations made by man in quartz veins with the gigantic mining operations of nature, we can only wonder that he has met with any of these large and rare masses of gold. The nugget argument, therefore, has no support, and falls to the ground.

" It is true, as has been asserted by the advocates of the chemical theory of the genesis of gold, that the fine gold in some places is of higher grade than that of the neighbouring quartz veins. This difference is, however, not marked, and something of this sort was inevitable, from the fact that all gold in quartz veins and placers is alloyed with silver. When minutely divided, and exposed to the action of chlorine and other agents which dissolve silver—so generally prevalent in the atmosphere and superficial deposits that silver is never found there in the metallic state,—some portion of the silver would naturally be removed from the surface of the particles of gold.

" It should also be said that the fine gold of placer deposits is usually purer than the coarse gold and nuggets, and the reason is, that the scales and fine grains of gold present more surface to be acted on by the agents which remove the silver. This process, which is essentially that called ' pickling ' by the jewellers, with the fact that gold itself does not

perfectly resist chemical agents, also affords an explanation of the frosted appearance which the surface of some placer-gold exhibits, and which has been hastily taken as proof that all such masses were chemically precipitated.

"Gold is usually said to be soluble only in aqua regia, and the idea is prevalent that it would remain for ever unaffected by any of the chemical agents in nature's laboratory. This is, however, far from true, as by the experiments of modern chemists it has been shown that it forms a great variety of chemical combinations. J. P. Pratt, epitomising the results of his experiments, says (*Journal de Pharmacie et de Chimie*, August 1870): 'Gold can be readily oxidised and salified by oxacids. There exists a liquid and volatile chloride of gold containing more chlorine than the sesqui-chloride. There exists, likewise, a sesqui-oxide and a carbonate of gold; and lastly, gold behaves in many instances like some of the other metals.' While all this is undoubtedly true, and it is also true that in the gossans of some of our Western mines gold exists in other than metallic form, these facts lend no confirmation to the theory of the chemical origin of placer nuggets. A more interesting fact, proving the solubility of gold by natural processes, is reported from the placers of California, namely, that the bark of some of the tree-trunks found buried in the blue gravel was largely replaced by iron-pyrites, which was rich in gold. Hence, we cannot deny that some gold has been deposited in the placers from solution; but this certainly does not include the nuggets and gold-dust. The facts which distinctly militate against that theory are briefly as follows:

"First. Deposits of placer-gold are always found adjacent to and lying below districts traversed by auriferous veins, and nowhere else.

"Second. The arcas where the quartz veins occur have certainly suffered great erosion, and mechanical forces have there been in action, tending to break down and comminute the quartz, and to liberate and wash the contained gold.

"Third. The conditions in which the placer-gold is found—namely, mingled with rolled fragments of quartz and in the irregularities of the surface of the bed-rock, where a washing process on a large scale has been in action, and where such washing process would have left it—prove the accumulations of gold to be mechanical rather than chemical. A deposit from chemical solution would not thus be circumstanced and localised.

"Fourth. The distribution of gold in placer deposits is also demonstrative of its mechanical origin; for in all cases known to me the nuggets and coarsest gold are found nearest the outcrops of the quartz veins that have supplied them; while the particles become gradually finer and finer as the line of drainage is followed from this point.

Hundreds of instances of this kind might be cited—enough, indeed, to form the basis of a mechanical law in itself an unanswerable argument to the chemical theory.

“Fifth. Nothing is more common than to find, in the placers, pebbles and fragments of gold-bearing quartz, which must have been derived from the neighbouring veins, and most of the nuggets have more or less quartz, just like that of the veins, still adhering to them.

“Sixth. The surfaces of nuggets almost always bear incontestable evidence to the battering they have sustained. They are generally rolled and rounded, and the surface is such as could be produced only by blows and friction. The cases where the surface of the nuggets is rough and frosted, as though from partial solution, are extremely rare, and afford no support to the chemical theory.

“Seventh. If the gold of placers were deposited from solution, we should necessarily find much of it crystallised, and forming strings and sheets running through the porous material ; whereas, as a matter of fact, crystals are never found in placer-gold, nor are sheets or threads. Scales, grains, pebble-like nodules, round, battered masses, these are what we find ; in other words, sand, gravel, and boulders of gold formed and transported by mechanical means.

“I cannot better illustrate the occurrence of gold in placers than by describing an isolated placer I have recently visited, which combines all the normal features of this class of deposits, and bears, in unmistakable characters, the record of its history. It is located at Osceola, Nev., and lies on the west flank of Mount Wheeler, said to be the highest mountain in the State. The central mass of Mount Wheeler is granite, but on its flanks occur upturned and metamorphosed Palæozoic rocks, quartzites, slates, and limestones. A spur, which extends northwards from the main peak, consists at its northern extremity of high, rough, and ragged masses of quartzite. These are succeeded on the south by a belt of talcose slate several miles in length, forming a ridge which rises 4000 ft. above Spring Valley, its western boundary. At its southern extremity, the slate belt is overlain by limestone. All these rocks are cut by veins of quartz, but those in the quartzite and limestone seem to be barren. The quartz veins in the slate belt are, however, numerous, and many of them are rich in gold.

“In the course of ages, the slate area has been extensively worn away, forming a ‘cirque,’ or semicircle, drained by several gulches, which combine below in one, and through which all the material removed from the mountain side has been spread over the slope to the valley below. In some places the mass of débris is 300 ft. or more in thickness ; above, it is narrowed between ridges of rim-rock, below spread out into a fan-shaped delta. The detrital material consists chiefly of rolled

boulders and pebbles of quartzite and vein-quartz mixed with sand and a cementing clay derived from the decomposition of the slate. All this contains gold; near the head of the gulch, that which is coarse; that below becoming finer and finer toward the valley.

"The climate of this region is now excessively dry, no water flowing through the gulch, except during a week or two in the spring, when the snows are melting on the mountain above. The temporary streams thus formed have cut narrow channels through the beds of gravel and boulders in the upper part of the gulch to the bed-rock below, only in the loose material. These modern channels have revealed the existence of others that are much older and broader, now filled, and in places deeply buried. These old channels would seem to have been produced at a time when the climate was more moist and the flow of water from the mountain much greater than now. As usual in such cases, the old channels are rich placer ground, the gold having accumulated in the depressions of the irregular bed-rock.

"The number of nuggets and the quantity of coarse gold taken from this placer is remarkable. The largest nugget found weighed 24 lb., and many others have been met with, weighing $\frac{1}{2}$ to 2 lb. At the time of my visit, two nuggets, weighing over 10 oz. each, were purchased, which had been found by one man on September 15th. Between \$200,000 and \$300,000 (40,000*l.* to 60,000*l.*) have been taken from this placer within the last 4 years, all by hand labour, and for the most part during the brief interval when a little water was flowing down the gulch. The nuggets have all been found toward the head of the gulch. Many of them have vein-quartz still adhering to them, and their derivation from the quartz veins which crop out on the mountain side above cannot be doubted. The whole of this placer has been purchased by parties who are about to bring water on to it from streams which drain the southern slope of Mount Wheeler. These streams run in gulches, which have the same general character with that at Osceola; but none of them contains gold, simply because the slopes they drain are not composed of auriferous rock.

"The significance of the facts I have given, and their bearing on the question discussed in the preceding pages, may be briefly summed up as follows:—Mount Wheeler is a very high mountain, which, though located in an exceedingly dry region, has through ages received on its summit sufficient precipitated moisture to form torrents that drain and have deeply scored its different sides. The material excavated is all spread in the vicinity, forming slopes of gravel and boulders at the mouths of all the gulches.

"In one of these gulches, the detritus is rich in gold; in all the others, it is barren. The rich one ~~has~~ received all the débris from a

portion of the mountain, composed of talcose slate, cut in every direction by auriferous quartz veins. The other gulches contain no gold, and there are no gold-veins at their heads. In the rich gulch, the gold becomes coarser and coarser toward the head as the outcrops of the quartz veins are approached. When to this is added that not a crystal of gold has been found in this placer, nor any gold that could be fairly considered a chemical deposit—but, on the contrary, all the masses are rolled and battered, often with adhering quartz—we have an array of evidence in favour of the mechanical deposition of the gold which cannot be gainsaid.

“Segregated Veins.

“Most of the quartz veins which carry gold belong to the class of what are called segregated veins. These occur only in metamorphic rocks, are lenticular sheets, limited in depth and lateral extension, and generally showing little of the banded structure so characteristic of fissure-veins. They consist mainly of quartz, in which the gold is sometimes free, but more commonly contained in iron-pyrites, with which yellow copper is often associated. Sometimes the gold is not strictly confined to the quartz veins, but extends more or less into the enclosing rocks, which are oftener than otherwise magnesian slates.

“The gold in segregated veins would seem to be indigenous to the formation in which it occurs, and not, as in fissure-veins, to have been derived from some foreign source. It is usually supposed that, before they were metamorphosed, the rocks which enclose the segregated veins contained gold generally, though sparsely, disseminated through them, and that, in the process of the segregation of the siliceous matter to form sheets of quartz, the gold was somehow gathered and concentrated by it.

“Sir Roderick Murchison, guided by his study of the gold deposits of the Ural Mountains, supposed that auriferous quartz veins were confined to Palæozoic rocks, but that the gold-impregnation had taken place at a comparatively recent date. It was demonstrated, however, by Professor Whitney, in the prosecution of the geological survey of California, that the metamorphic slates which carry gold in the Sierra Nevada are of Triassic and Jurassic age; and in the light of later observations, we may say that metamorphic rocks of all ages contain auriferous veins. Nearly all the great mountain chains of the world contain more or less of such veins, and as these mountain chains have been the great condensers of moisture, and erosion has been constantly wearing down their slopes, placer deposits have been formed which have supplied most of the gold yielded by the earth to man. As it can be procured from them by the simplest methods, the work of its extraction was begun by prehistoric races, and the Altai, the Himálayas, the Ural Mountains, the Australian

Alps, the Sierra Nevada, and the Rocky Mountains have in turn contributed their millions to the treasures of the world. These mountain chains are of very different ages, and we have abundant evidence that gold has existed in some of them from the earliest geological times. The oldest mountains of which we have any knowledge—the Laurentian, of Canada, now nearly removed by erosion—contained auriferous quartz veins that have supplied gold to all the successive formations which have been derived from their ruins. The gold-impregnation of the Laurentian rocks dates back certainly to the period of their metamorphism; and this was pre-Silurian, for the undisturbed Lower Silurian strata overlap and partially cover these gold-bearing rocks.

“In the same way, the gold at the Black Hills is proved to be pre-Silurian, since the Potsdam sandstone which abuts against the Archean nucleus of the hills in places contains rolled fragments of the Archean rocks, and gold washed from them in such abundance as to form rich mining ground—the so-called cement deposits of that region. The distribution of gold from the Archean rocks has probably been constantly going on from the Silurian age to the present day. This is shown in the almost universal dissemination of gold through the drift of New England, New York, Ohio, &c., where the superficial materials have been largely derived from the Canadian highlands. In Ohio, gold is found in the drift clays, sands, and gravels, and locally in as great quantity as in the poorer placers of California. There is little doubt that the mechanical sediments derived from the wear of the Archean rocks all contain gold, and since it has been proved that gold exists in sea-water, it has probably impregnated all the organic marine sedimentary rocks as well. In the subsequent metamorphism of some of these strata it has been concentrated in such a way as to produce auriferous quartz veins rich enough to be worked.

“From these facts it will be seen that there is no geological age which can be called the age of gold. It existed in the oldest rocks known, and from them and their derivatives, more modern rocks, it has been, and is now being, constantly distributed by both mechanical and chemical processes. Even some of the igneous rocks of the Western country are said to contain minute quantities of gold; and this is not surprising, if, as is supposed, much of our volcanic material is a fused condition of sedimentary rocks.

“Gold in Fissure-veins.

“As is well known, gold is a frequent constituent of the fissure-veins of the Far West. The ore of the Comstock vein has yielded about 47 per cent. of gold and 53 per cent. of silver; and it is probable that

one-half of the so-called silver veins contain gold in sufficient quantity to be of practical value. In some true fissure-veins, gold is the only valuable ingredient, but more generally it is associated with several other metals. The Revenue mine, at Vuscarora, Nevada, contains silver in the form of arsenical and antimonial sulphide, and gold in iron-pyrites, frequently crystallised, lining cavities. At Eureka, the ore occurs in chambers, which were originally filled from a solution issuing through fissures from below, and deposited as argentiferous galena and auriferous pyrites, the silver and gold being in nearly equal proportions. In the great veins of Bingham Cañon, and at the Cave mine, near Frisco, in Utah, the combination is the same, and as at Eureka, the sulphides have been decomposed to a spongy, rusty gossan. At the Bassick mine, in Colorado, gold exists free, or in combination with tellurium, and associated with zinc, copper, and iron. In all these, and many other cases which might be cited, the gold has been brought up in a hot solution impregnated with mineral matter far below, and deposited as the temperature and pressure were reduced. The formation of this class of auriferous deposit is well illustrated by the Steamboat Spring, in Western Nevada, where hot water, flowing out through fissures produced by subterranean forces, is depositing a siliceous veinstone, containing sulphides of iron, copper, oxide of manganese, and metallic gold. There is little doubt that, in the great mineral belt lying between the Sierra Nevada and the Rocky Mountains, where, in Tertiary times, volcanic activity was exhibited on a grand scale—sedimentary rocks upheaved, and fissured in every direction, with great outflows of fused material—hot springs, like the Steamboat, were everywhere busy, doing similar work. Bursting out at different places and times, and flowing from different sources, the solutions they carried and the ores they deposited varied greatly; but the methods of accumulation, transportation, and deposition were essentially the same, namely, the leaching of various rocks by steam and hot water under great pressure, by which silica and sparsely disseminated metals were gathered and driven toward the surface, to be deposited as the pressure and temperature were reduced. Gold collected in this manner was unquestionably taken into chemical solution, and in the resulting vein deposits we find it in strings, scales, and irregular masses, often beautifully crystallised and associated with other crystallised minerals which are certainly chemical precipitates.

“We may sum up the teachings of geology in regard to the genesis and distribution of gold by saying:—

“First. Gold exists in the oldest known rocks, and has been thence distributed through all strata derived from them.

“Second. In the metamorphosis of these derived rocks it has been

concentrated into segregated quartz veins by some process not yet understood.

“Third. It is a constituent of fissure-veins of all geological ages, where it has been deposited from hot chemical solutions, which have leached deeply-buried rocks of various kinds, gathering from them gold with other metallic minerals.

“Fourth. By the erosion of strata containing auriferous veins, segregated or fissure, gold has been accumulated by mechanical agents in placer deposits, economically the most important of all the sources of gold.”

According to W. H. Penning (1888), “Gold occurs under entirely novel conditions in each of the main auriferous tracts of the Transvaal, and equally so in other parts of South Africa.

“The Kaap Valley goldfields are situated upon and around the margin of the low country, the rocks of which are schists and shales, with interstratified beds of quartz-rock and quartzite, which are often highly auriferous. The rocks having been tilted from a horizontal into a nearly vertical position, these gold-bearing beds simulate true fissure-veins, but it is beyond question that they are stratified deposits; therefore, auriferous beds, such as were not known before (so far as I am aware), although ‘flat bands of auriferous quartz have been discovered in dykes of diorite’ (Phillips), which, however, being in diorite, are veins, and not beds at all.

“The Lydenburg goldfields are upon the plateau, where the rocks are horizontal shales, sandstones, and cherts, with inter-bedded seams of auriferous quartz, and crystalline conglomerate, and exhibiting here and there the peculiar sandy deposits, often very rich in fine gold, locally known as ‘rotten-reefs.’

“In both of the goldfields there are occasional true fissure-veins and stringers of auriferous quartz, but these form the rather rare exceptions to the general rule.

“The Witwatersrand goldfields are upon the western edge of the High Veldt plateau, where we find a series of rocks that present some decided and important differences from those of both the other goldfields. These are chiefly soft sandstones, with some shales, cherts, and quartzites, like those of the High Veldt proper, but tilted, as are the rocks of the low country, into a nearly vertical position.

“At intervals between the sandstones and other beds, there are interstratified layers of conglomerate, which vary in thickness from a few inches up to several feet—always following the same line of strike as the beds in which they occur, and having the same dip or inclination. These conglomerates are of all degrees of coarseness, from a large-grained sandstone with occasional pebbles of small size, up to a regular

'pudding-stone' with pebbles as large as hens' eggs, or even of greater size. They vary also in compactness and colour, some being soft, others hard, some white, others grey, purple, or brown, the inclosed pebbles being equally variable from white quartz to black hornstone. From its peculiar appearance, the Dutch have given this conglomerate the name of 'Banket' (pronounced *bahnkett*), or almond-rock, which is very expressive, and may be conveniently retained, as it is in general use on the Fields. This rock is auriferous, sometimes highly so, and in very few instances, in this particular region, is it quite bare of gold.

"There is a striking similarity in the present position of the auriferous seams of the Kaap Valley and the Witwatersrand, both being broken through and tilted up by a central mass of granite, but the resemblance extends no further. It may be worth while to briefly describe this feature, as indicating where other discoveries are certain to be made, and I may quote the words used by me in a paper read before the Geological Society in 1883.

"An intrusive plutonic rock, geologically newer than the stratified rocks, but still in an inferior position, occupies the whole of the lower ground of the western part of the Kaap Valley. It resembles coarse granite, and consists of quartz and felspar, with but little, if indeed any, mica in its composition. Although this granite (for such it may be called) forms generally the lower ground of the head of the valley, it rises into hills and ridges, some of which have a considerable elevation. In the Kaap Valley the surface of the granite forms an ellipse about seventeen miles in length by ten miles in width, with a narrower prolongation in a northerly direction. This mass of granite represents a great centre of plutonic upheaval, of course posterior to the period of the Kaap Valley rocks, which it has greatly tilted all round its margin; it was, however, anterior to the deposition of the rocks of the overlying unconformable formation, as they are still nearly horizontal. Nearly all around the granite centre is a series of rocks which have been tilted by it into a more or less vertical position.'

"In the case of the Witwatersrand, the granite, which is of precisely the same character, occupies the ground from the foot of the steep northern escarpment overlooking the head-waters of the Limpopo which traverse the lower area between it and Pretoria. The width of the granite is about 25 miles on a northerly line from the Witwatersrand, the stratified rocks coming on again about 6 miles south of that town, which is 33 miles or so from Johannesburg.

"The rocks of the Rand are chiefly sandstones, sometimes micaceous, with occasional shales, cherts, and quartzites, especially towards the base of the formation which abuts against the mass of granite already mentioned. The intrusion of this granite has tilted the rocks to a high

angle near its northern margin, where they are almost vertical; farther south they gradually assume a less inclination, and eventually are seen in a nearly horizontal position.

"A similar mass of granite occurs at and near the village of Klerksdorp, about 25 miles west of Potchefstroom, appearing to follow the valley in a north-westerly direction towards the Maqwassie Mountains; it has also tilted up the stratified beds, and exposed on both sides of the Schoen-Spruit the outcrop of "bankets" similar to those of the Rand. At the time of my last visit, in July 1887, this new field was beginning to attract attention.

"In all these cases, the dip of the stratified rocks is away from the central granite, and their outcrop follows its margin in more or less parallel lines, except where intercepted by intrusive dykes of later date, or broken by faults, some of which, on a rather large scale, occur in parts of the Kaap Valley.

"The practical question naturally arises whether these auriferous seams possess any great lateral and vertical extension. The former can be readily proved, but the latter is not so easy of demonstration, although, in my opinion, the one virtually hinges on the other. To take the bankets as an illustration, there is no *à priori* reason why these gravels should have originally extended farther in an east and west than in a north and south direction. If it were so, it would have been highly improbable, almost impossible, that the upheaval by the granite should have exposed their present edges exactly along what was, on this presumption, an attenuated line of deposition. Such line would surely have been tortuous, not straight, and the outcrop of the conglomerates at the present surface would in consequence have been intermittent. But with very few exceptions it is decidedly continuous, and it may therefore be fairly argued that the bankets originally extended, and still extend, as far in the one direction as in the other. That is to say, if they can be traced, as they undoubtedly can be, for 30 or 40 miles, with a practically unbroken outcrop, from east to west, there is no valid reason why they should not have extended 30 or 40 miles from north to south. If so—and the present exposed edge represents a medial, or what may be called an average line—they pass beneath the surface of the ground for half that distance, not vertically to any great depth, for I have shown that the strata only where near to the granite are almost vertical, and that they have a gradually lessening dip, which, in the course of 3 or 4 miles, must bring them back into their original horizontal position.

"These reefs can be traced on the surface for 30 miles at least, and the fact is too well known to require confirmation; it may, however, be mentioned that the tracing of the so-called Main Reef was such an easy matter to prospectors, that its apparently sudden ending near the

western boundary of the farm 'Driefontein' was a source of surprise—not to say of consternation. It was, however, lost there, and considerable sums of money have been forthcoming for its re-discovery—no very difficult matter, as its disappearance at or about that point is due to its being 'thrown' by a 'fault,' and not to its actual termination.

"Again, the so-called Kimberley, Sunday, and Free State reefs are known from end to end of the Rand; this group being distinguishable, not only by its position relative to the Main Reef, but by its own well-defined characteristics. And all the reefs, in whatever group, maintain a fairly consistent width or thickness, varying somewhat, but to no great extent, and (of course, with some few exceptions) not 'pinching out' entirely, or their outcrop would fail to be, as stated above, unbroken for 30 miles. There are some few instances in which these reefs really die out, both laterally and vertically, as must of necessity be the case in deposits that were once horizontal, and whose extent was bounded by ever-varying conditions. But these lines of limitation would have been so far apart, that the chance of the occurrence of more than one such line (i.e. of the pinching out of more than one reef) is reduced to a minimum, within the breadth and depth of even an extensive series of mining operations. If occasionally a reef should be found to thin entirely away at a depth, there is an exactly equal probability of another coming in that had not been exposed at the surface, as is evident from a consideration of their mode of origin.

"This argument in favour of the downward extension of the gold-bearing 'bankets' applies with even greater force to the Kaap Valley auriferous deposits, the outcrop of which extends more than half-way round a nearly circular mass of granite, than to the 'bankets' which follow a line curved only about 3 miles from the straight in its known length of 30 or 40 miles. It bears also upon the still horizontal quartz seams of the Lydenburg goldfields, the outcrops of which may be greatly obscured, but some have been traced intermittently for considerable distances, and along tortuous contours, which fact points to the conclusion of their extending over large areas.

"Another point of great practical importance is the origin of the gold in these peculiar deposits; referring not so much to the source whence it was actually derived as to the mode by which it was conveyed. I am aware that this is approaching debateable ground, but, as much depends thereon, I would offer a few remarks which may perhaps assist in solving the question whether we may expect to find an equal or a partial distribution—in other words, whether the reefs will be found 'patchy' or their yield fairly persistent.

"It will be convenient to again take the 'bankets' as presenting the clearest evidence, for whatever might be said of the vertical quartz-rocks,

there can be no question that these conglomerates, full of water-worn pebbles, are alluvial deposits. But the gold they enclose is sharp reef gold—not water-worn as nuggets are supposed to be, but as I believe very few are—therefore not alluvial ; neither has it been (except in some instances to be again referred to) formed into nuggets by some secondary process of aggregation.

“It is, perhaps, not easy to conceive by what agency masses of gravel can have been accumulated that are several miles in length and probably of equal width—such gravel being of fairly uniform thickness throughout although in some cases with considerable variation. The conditions of deposit were intermittent, for we find numerous layers of conglomerate, some only a few inches, and many several feet thick, with uniform beds of sandstone between. Sometimes a little talcose shale intervenes, but this appears to have been in very limited areas, as it soon thins out in each direction. There are also some calcareous deposits, sufficiently persistent to be supposed to mark an horizon ; for instance, they are found in connection with the ‘Kimberley’ series of conglomerates. But the rock generally associated with the gravels is sandstone, varying in coarseness and hardness, with foliated schistose rocks below them, then cherts and quartzites at the base of the formation which abuts against the granite. By whatever agency the gravels may have been accumulated over such large areas, we have ample evidence of their existence in the upturned edges of the ‘bankets’ of the Witwatersrand, at least as regards their length of many miles, and that they possess great breadth also is, as I have endeavoured to show, a reasonable conclusion.

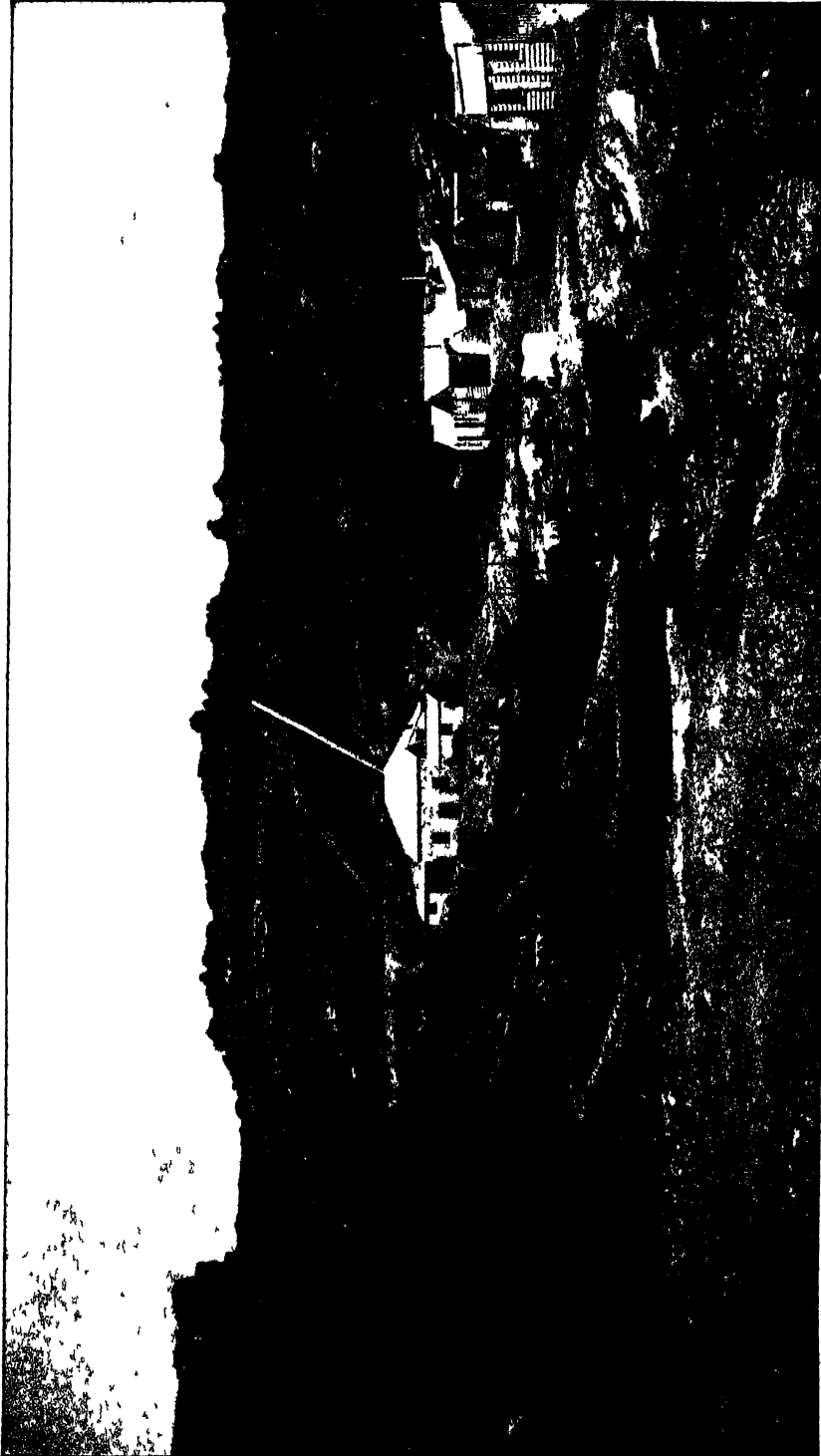
“If the gold were alluvial, brought into the gravels in the same manner as the pebbles must have been—by the action of running water—it would certainly be found in ‘leads,’ and probably the larger portion of it accompanying the heavier pebbles. But thus far it appears to be equally distributed, for each ‘banket,’ although it may differ in auriferous value from those above and below, maintains an equable richness for long distances. The conglomerate, as we have seen, was spread with remarkable regularity over a large area, and we are justified in concluding that, as in so many points within that area, the conglomerate is rich in gold, so it will be in the intervening portions, because there is no evident reason why the agencies of gold distribution should not have been equally uniform. It is contended that, in the absence of any evidence to the contrary, it is right to assume that the ‘banket’ is no less rich in the intervals between those points ; that, in short, the opened ground is a fair indication of that which is undeveloped. Looking to the sharp form of the grains of this gold, to its freedom from the coating so often found on that which is really alluvial, and to the equality of its distribution, I am convinced that this was deposited—at the same

time that the gravels were being accumulated—from water holding gold in solution.

“It seems to me that this is the only way in which to account for the presence of gold over such large areas as these must have been, with so little variation in its distribution. There is variation vertically, the term being, of course, applied to the deposits in their original position consequent on the varying rapidity of the formation of gravel and of the deposition of gold. In this way, not only would one ‘banket’ differ from another in its proportion of gold, but its own component layers would exhibit unequal degrees of richness. This is generally the case; indeed I have seen some which are very rich in coarse gold on the one side, and contain but a mere fractional part of fine gold on the other. This variation is also noticeable in some of the quartzose rocks of the Kaap Valley, especially one in regard to which I remarked, in the paper alluded to above:—‘One side of the quartz seam called Moodie’s Reef, which would be its north wall if a true vein, but is what was originally its underside as a bed, is plentifully sprinkled with specks of gold, rather fine, but still visible to the naked eye. There is gold also in the body of the quartz, but, so far as I have been able to ascertain, only in small quantity.’ Whether or not my view be correct as to the origin of the gold in these stratified deposits, I venture to believe that the evidence adduced, founded mainly on my own personal observations, justifies the statement that not only in two of the three distinct auriferous regions of the Transvaal, but in all of them, the gold exhibits decidedly novel modes of occurrence.

“If this be so, if the gold be actually in beds—and I have myself no doubt whatever upon the point—is it not a feature of immense importance to South Africa and the world at large? We have here not mere fissure-veins which may rapidly disappear, but stratified deposits (inclosing the auriferous layers) of enormous area, which have hitherto been merely scratched upon their outcrop—for what are a few hundred feet when miles are in question? Not useless miles straight down, as might appear from some of those exposures where the beds are seen at the vertical part of the curve made by the granite in breaking through them, but miles of deposits gradually flattening outwards from the centre of upheaval. Those nearest to the granite may soon get below a workable depth, but not so with those that form the outer concentric rings of outcrop around the granite. As we have seen, in the Witwatersrand at any rate, these outer beds dip away at an angle of only a few degrees; so that in these, and in the horizontal seams of the Lydenburg Fields, the quantity of available auriferous ore, hidden though it be under newer deposits, is practically unlimited, and certainly beyond estimation.

“It may be mentioned here that, although the auriferous ‘banket



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somewhat resembles the 'cement-stone' found in many parts of the world, it is not by any means the same thing. There are two kinds of deposit known by that name, through their being consolidated or cemented together by a matrix of ferruginous sand ; one is old auriferous gravel, well known in Australia, California, and South America ; but the gold it incloses is waterworn and strictly alluvial. The other is rather common in South Africa, and is merely an old deposit of surface-wash, which is auriferous where the conditions are favourable. It has formed generally on large flats, and it rests indiscriminately on the solid rock or on true river gravels, as at the town of Lydenburg in the Transvaal. There is gold in the neighbouring hills, and it is found in alluvial form both in the river gravel and the overlying cement-stones, but in both having a very unequal distribution."

GEOLOGICAL AGE.

In attempting a classification according to geological age of the gold-deposits now known to exist, the statements of various authors have to be taken at what they may be worth, and it is quite possible that future exploration in some fields will modify present views. For instance, it is often assumed that when veins cease upwards of the line of contact of two formations this is a proof that they existed before the deposition of the superior formation, which is by no means always assured. Again, in reference to the volcanic rocks associated with gold, it must be borne in mind that scarcely two geologists apply exactly the same name to the same rock ; indeed, it becomes only too obvious that the present system of geological nomenclature is no system at all. With these facts in view, this section must be considered in some respects as tentative only, and it is to be hoped that its incompleteness will incite all interested to furnish more exact details of what they observe in future. The classification adopted is as follows :—

Metamorphic	Triassic
Laurentian	Jurassic
Cambrian	Cretaceous
Silurian	Tertiary, Miocene
Devonian	Tertiary, Post-Miocene
Carboniferous	Igneous.

METAMORPHIC.—"Metamorphic limestone" is stated by Probert to be the country-rock of the celebrated Richmond mine, in Nevada. Russians ascribe the Siberian alluvial gold to the ferruginous quartz of the metamorphic schists. In Hayti, every stream running through the metamorphic rocks in the immediate vicinity of masses of syenite carries gold, while those running exclusively in syenites, or at a great distance

from them, are without the precious metal. The gold of India is mainly derived from quartz reefs traversing metamorphic and sub-metamorphic rocks, and occasionally from gneiss, chloritic schists, and quartzites showing no quartz. The placer gold of Yesso, Japan, is derived from metamorphic strata. The mountains forming the watershed between the Chirchik, Tersi, and Talas rivers, in Siberia, all of which carry gold, are mainly composed of metamorphic schists, veined with diorite, syenite, and quartz. The prevailing formations of the Yeniseisk gold-fields, Siberia, are metamorphic schists, clay-slates predominating. The gold of the Trans-Caucasus is of metamorphic origin. That of New Caledonia is found in mica-schists.

According to Prof. Newton, the gold-ledges in the schists and slates of the Black Hills of Dakota are of Archæan age (Archæan being the term founded by Dana for the older non-fossiliferous strata, mostly composed of crystalline and metamorphic rocks—granite, syenite, gneiss, and micaceous, talcose, hornblendic, and chloritic rocks), and were formed during the folding of the metamorphic rocks. The whole area of the gold-field in the Black Hills was, at the time of the upheaval of the range, covered by the Potsdam [Cambrian: Lower Silurian] and subsequent formations. It is probable that the Potsdam conglomerate, formed by the primary erosion of the metamorphic rocks and their enclosed auriferous quartz ledges, contained considerable quantities of gold, and by the disintegration and denudation of this conglomerate since the elevation of the Hills, the gold which it contained has been set free and concentrated anew in the placer gravels. This may in part account for the richness of some of the older and more elevated gravel deposits along the valleys of the present streams.

Jenney considers the metamorphic rocks of the Black Hills as separable into two distinct groups, whose lithological characters are marked and persistent: a western series, or group of schists, and an eastern series, or group of slates, the line of separation being only imperfectly indicated. The western series consists of quartzose schist and garnetiferous, quartzose, and ferruginous mica-schists, together with some gneiss, chloritic and talcose (or hydrous mica-) schists, hornblendic schist, and quartzite. The whole series is coarse in texture and highly crystalline, and it contains many seams or veins of quartz traversing the schists conformably with the stratification, and having usually a swelling or lenticular structure. These veins are interlaminated, and are not often of any great width; they contain finely disseminated gold, and have probably afforded by their disintegration the larger portion of the gold found in the valleys and gulches. The granite masses are wholly within the area of the schistose rocks. The eastern series is composed of metamorphic rocks, distinguished from the western mainly by their exceedingly fine and compact texture, though, as shown by Caswell,

their ultimate mineral composition is quite similar. The rocks are mainly micaceous clay-slate, clay-slate, silicious slate, and quartzite. The last forms persistent strata, 50 to 200 and sometimes 500 ft. in thickness, which may often be traced for long distances with little variation. The quartzite frequently contains seams or veins of interlaminated or ribbon quartz, and with them are associated large deposits of hematite or specular iron-ore, also interlaminated with quartz. Frequently the quartz seams are highly ferruginous, and in places they have been found to contain undecomposed pyrites. Unquestionably they are often auriferous. In many instances, the quartz veins are undoubtedly auriferous, and the larger portion of the gold found in the gravels of the Hills has originated from them; but by reason of its sparse dissemination in the vein-matter, specimens containing visible particles are not often met with. The very fine state in which the gold is found in the gravels of the southern end of the Hills is also an evidence of the fine state of division in which it occurs in the veins. The gold encountered in the ancient gravels of the Potsdam formation must be referred, together with that obtained from gravels of the most recent formation, to the quartzes of the Archæan as its source.

In some of the valleys in the northern end of the Hills, where rich deposits of placer-gold have been found, the bed-rock of the stream is Potsdam sandstone. The auriferous gravels may in such cases have been derived directly from the wearing down of the schists and slates on the upper courses of the creeks; but it is more probable that they came indirectly from the same source through the medium of the Potsdam. The streams which drain the Archæan area are concentrating the gold by a method not very dissimilar from that of the Potsdam waves, and it is reasonable to suppose that dirt which has been first rocked by the waves and then sluiced by the creeks will hold its gold in a more concentrated condition than that which has had the benefit of but one process.

Foote divides the metamorphic rocks of the known auriferous area of India into 3 groups, distinguished by the local names of Dhoni, Kapatgode, and Surtur, while two other groups, the Gulduck and Mulgund, are apparently barren. Quartz reefs occur in all the series; but according to the natives, it is only streams draining the Surtur series that carry gold, and these lie entirely in an area of chlorite-schists and diorite.

LAURENTIAN.—E. McCarthy (Min. Jl., July 1st, 1882), speaking of the three formations in which gold occurs on the African Gold Coast, mentions quartz reefs traversing the older rocks of clay and talcose slates and schists, upon which the second series of rocks (sandstones and conglomerates, often so metamorphosed as to pass into gneiss) are laid, and in almost every case they are auriferous, while the more recent quartz reefs traversing the second series of rocks, and sometimes even the

gravel beds, are non-auriferous. The older rocks he considers may be of Laurentian and Cambrian age. Williamson likens the auriferous rocks of Parahyba, Brazil, to the Laurentian of Canada.

CAMBRIAN.—Notwithstanding all that has been published by the various geologists who have studied the gold-region of Minas Geraes, Hartt considers that the exact succession of the different members of the metamorphic series lying just inside of the gneiss belt has never been satisfactorily worked out. The clay and talcose schists, the itacolumite, itabirite, and other associated metamorphic rocks of this region appear to him to be Lower Palæozoic in age. He has called attention to the striking resemblance borne by the clay-slates and associated quartzites to the gold-bearing rocks of Nova Scotia, and has suggested that they may be the equivalents of the Quebec group of North America. The gold-bearing rocks in Minas Geraes resemble the similar auriferous series of the southern Atlantic States, in which itacolumite occurs. Clay-slates with auriferous veins occur in other parts of Brazil besides Minas, as, for instance, in Goyaz, and in the vicinity of Cuiabá in Matto Grosso. These rocks are everywhere so metamorphosed, that all traces of fossils have been completely obliterated.

According to Selwyn, the gold-bearing series in Nova Scotia resembles the Cambrian and Lingula-flag series of N. Wales.

SILURIAN.—It is quite unnecessary to insist upon the gold-yielding capabilities of the strata of Silurian age, so long thought to be the only auriferous formation. Though its claim to a monopoly of gold-veins is now quite disproved, it maintains the first rank in point of importance in that respect. Ulrich remarks that in Victoria, the Silurian, as a whole, is the most important rock-formation to the gold-miner, on account of its containing the matrix of the gold in the countless number of veins, lodes, or reefs of quartz that traverse it—the number of those actually proved gold-bearing amounting already, according to Brough Smyth's Mining Statistics of 30th June 1874, to 3367, and being still steadily on the increase, through the untiring prospecting energy of the miners. These reefs vary from less than 1 in. to above 100 ft. in thickness, whilst their longitudinal extent ranges from less than 100 ft. to several miles. Their mean strike is, in the Lower Silurian series, generally conformable to that of the strata; reefs that cross the latter being very scarce (Taradale and Stawell are the chief localities where cross reefs occur—one at the latter place being celebrated for its richness in gold). In the Upper Silurian are more exceptions to this rule; and it must also be remarked that this series is not by far as rich in genuine quartz-reefs, and those found—though on the average, perhaps, richer in gold per ton of stone—are not as thick and persistent as those of the lower series. The greater amount of quartz-gold produced by Upper Silurian

districts comes from irregular blocks and veins that traverse, or are closely associated with, dykes of diorite-greenstone. Regarding the dip or underlie of the reefs, it varies at all angles between nearly horizontal and vertical, and coincides not only in most cases in direction with the dip of the strata, i. e. either east or west, but its angle is often so nearly identical with that of the latter as to impart to the reefs the appearance of interstratified deposits. If it were not for the frequency of lateral branches—"leaders"—joining the main bodies at all angles, both at their hanging- and foot-walls, they would in reality have to be considered as "layers" or interbedded deposits; but, as it is, they represent in these cases so-called "layer- or bedded lodes."

Another feature of importance is, that many reefs show, from the surface downwards, an endlong dip, or a dip in strike either north or south, sometimes in both directions, whence follows an expansion in depth. Prominent amongst the former are the so-called "block reefs," i. e. reefs which in their longitudinal extent show frequent contractions, consisting, as it were, of a series of—in horizontal section—lenticular-shaped blocks, frequently of considerable thickness in the centre, which are connected at longer or shorter distances, sometimes by thin veins of quartz, sometimes by ferruginous clay-casings only, and each block dipping from its outcrop at a certain angle—rarely at right-angles—to the line of strike either north or south, i. e. the blocks of one and the same reef, and generally of all the reefs in a district, show invariably the same direction of dip—northward being the most frequent. The expression, often used by Victorian miners, "that it requires a shaft of a certain depth, either north or south of an outcrop, to strike the 'cap' of a reef," refers to this feature, and will thus be easily understood. A peculiar and interesting occurrence are the so-called "saddle reefs," described in another section. As the term indicates, a reef of this class presents, in east and west section, the shape of a saddle, i. e. a generally very thick central part with a certain flat dip in strike, either north or south, bends—whilst decreasing in thickness—sharply downward on either side; and of these lateral parts, the so-called eastern and western legs, one dips generally with the strata, whilst the other opposite one crosses them.

The mode of occurrence of the gold in reefs can be brought under the following main heads:—

1. The metal is equally distributed throughout the whole thickness and extent of the reef. This seems to be the rarest occurrence. Generally, richer and poorer places alternate, especially where the thickness of the reef changes, the contracted portions being mostly richer than the wider ones; but the contrary having also been found to be the case in a marked manner in some reefs, no rule can be established in this respect.

2. It exists in irregular larger and smaller patches, both in strike and dip, throughout the reef. More frequent than the former.

3. It occurs most frequently in so-called "shoots," i. e. in stripe or band-like areas or portions of the reef of various widths, which dip at various angles, either north or south in strike; rarely at right-angles to the latter. Both the width and angle of dip of the shoots vary in each individual case, generally but little, allowing thus tolerably accurate calculations to be made as to the depth at which a shaft, for instance, situated some distance from its outcrop, ought to strike such a shoot in the direction of its dip.

4. A not uncommon case, especially in strong reefs, is that the auriferous portions, either irregular patches or shoots, occupy only a certain width along the hanging- or foot-walls, more rarely along both walls, whilst the remainder of the reef is barren. Some exceptional cases have also been observed, however, of the gold being confined to a certain width in the centre of strong reefs, the quartz along both walls proving worthless. Most of these occurrences form structural features of the respective reefs, inasmuch as the gold-bearing portions represent separate bands, divided by thin black casings from the rest; they appear, in fact, different in date of formation from the latter, either older or newer, thus indicating a re-opening and successive filling of the reef fissures.

Touching the depth to which the gold extends in Victorian reefs, the hypothesis advanced by high authority in the early days of the gold-fields, namely, "that the metal would be found to quickly decrease in quantity, and entirely to disappear at a limited depth," has, fortunately for the prosperity of the quartz-mining industry, proved, in the main, incorrect. For depths of between 500 and over 1000 ft., at which payable stone is being worked in many mines already, and with no unfavourable signs, even in the deepest, of its termination lower down, can surely not be called limited. That the quartz becomes, on the average, poorer in gold with increasing depth from the surface seems, however, to be the case in a great number of the deep mines, though exceptions of an actual improvement are not wanting. In fact, in many deep mines, profitable working is at present only carried on by great economy and improvements in the system of mining and the machinery, especially that for crushing the quartz, combined with the gold-saving appliances; and last, though not least, by the saving and treatment of the auriferous pyrites—iron-, arsenical, and magnetic pyrites,—ores which generally appear at or beneath the water-level, and seem to increase in quantity in depth, whilst the free gold gradually decreases.

The yield of gold directly and indirectly drawn from Upper and Lower Silurian rocks in Victoria has reached approximately 60,000,000 oz., value 250,000,000/. Pegler places the gold area of French Guiana in Silurian. Most or all of that found in the Andes would seem to be of similar age.

Probably the whole of the Siberian gold is derived from the Silurian limestones and schists, which are penetrated and in some places metamorphosed by granites, porphyries, serpentine and diorite. Part of the New Zealand gold-fields are in Lower Silurian hornblende rocks, and part in Upper Silurian Batow river slates. Many of the Queensland reefs are in Silurian metamorphic rocks.

DEVONIAN.—The gold-workings of the Devonian areas of North Gippsland have been ably described by A. W. Howitt. The localities in which the auriferous deposits are being worked are as follows:—(1) Lower Boggy Creek; (2) The Mitchell river, between Tabberabbera and Lindenow Flat; (3) Maximilian Creek.

It is a question of no less moment from an economic than of interest from a scientific point of view whether the gold found and worked in the "claims" of the above localities is of local or extraneous origin. If it is possible to show reasons for the belief that the gold is local and not derived from the waste of Silurian areas, then an inference is justified that similar auriferous deposits may exist, and may be discovered, throughout these formations, at a distance from the older Palæozoic rocks.

Howitt gives the following table of assays of alluvial gold from the above localities. The assays were carefully made before the blowpipe, and each one consists of the mean of two or more nearly accordant assays of the same gold.

COMPARATIVE TABLE SHOWING THE FINENESS OF ALLUVIAL GOLD FROM VARIOUS MIDDLE AND UPPER DEVONIAN LOCALITIES IN NORTH GIPPSLAND.

No.	Au.	Ag.	Oxidisable Metals and Loss.	Locality.	Geological Formation.	Character of Gold
1	94 950	4·850	·200	Lower Boggy Creek ..	Upper Devonian, resting upon trappean porphyries.	Fine, laminated, and scaly.
2	96·137	3·378	·485	Tabberabbera, 4 miles above Wentworth river.	Middle Devonian, with Upper Devonian outliers.	Fine, scaly.
3	96·322	3·668	·010	Ditto, 3 miles above Wentworth river.	Ditto	Ditto.
4	96·340	3·660	..	Ditto, junction of Wentworth and Mitchell rivers.	Ditto	Ditto.
5	97·174	2·174	·652	Ditto, 2 miles below Wentworth river.	Middle Devonian, overlaid by Upper Devonian.	Scaly, a few small quartz specimens.
6	97·070	2·386	·544	Mitchell river, above Glenalladale.	Upper Devonian ..	Laminated and scaly.
7	83·872	16·128	..	Maximilian Creek, Upper Gladstone.	Ditto, resting on older rocks.	Nuggety and rounded.
8	84·214	15·571	·215	Ditto, Lower Gladstone	Upper Devonian ..	Ditto.

1. Lower Boggy Creek.—This stream, after leaving the Silurian auriferous quartz-bearing strata of its upper course, enters a tract where the bed-rock is a peculiar quartz-porphyry, and the only overlying strata, exclusive of marine beds of Upper Tertiary (Pliocene) age, belong to the Upper Devonian group. The gold found in Lower Boggy Creek differs completely in its physical character from that met with where the upper part of the stream flows over Silurian rocks. It is finely laminated or scaly, while that of the upper creek is coarse and nuggety.

In considering this question, it seemed to Howitt, from the facts with which he was then acquainted, that the gold found in Lower Boggy Creek might have been derived from the wearing down and rearrangement—in fact the “ground-slucing”—of great accumulations of waste from the Silurian hills, when the later Tertiary coast-line extended far up the valley of Boggy Creek. As already noted, it is worthy of remark that the physical character of the gold found at Lower Boggy Creek differs very greatly from that of the Upper Silurian, and it is to be added that the change in character is coincident with the change in the formations, and that it entirely accords with the character of the gold found throughout the gold-workings in rocks of the Devonian age of the Mitchell from Tabberabbera downwards. These and other facts which will be shown in respect to other localities may require this hypothesis to be somewhat modified, by adding to the auriferous materials also the waste of the Iguana Creek beds themselves.

2. Mitchell River.—Gold workings exist at the Mitchell river from some miles above Tabberabbera down to within 2 miles of the commencement of the wide valley of Lindenow Flat. The claims are all worked on the banks, on the points of the spurs, and, when the water is sufficiently low, in the river-bed. But they are much scattered, and, owing to the great difficulty experienced in constructing water-races for sluicing, the extraction of the gold is mostly performed by the primitive method of “cradling,” and consequently does not remunerate miners as it otherwise would do. So far as it is possible to judge from the few claims opened, and from the surface indications seen elsewhere, there can be but little doubt that the whole extent of river mentioned is more or less auriferous.

This extent may be divided into three portions :—

- a. Middle Devonian (Tabberabbera shales), with mere traces of the Upper Devonian.
- b. Middle Devonian (Tabberabbera shales), overlaid by a great thickness of Upper Devonian (Iguana Creek beds); and
- c. Upper Devonian (Iguana Creek beds).

The first portion (a) is situated about Tabberabbera, being perhaps 4 miles above and the same distance below the junction of the rivers

Mitchell and Wentworth. The character and fineness of the gold found will be seen from the tabulated form given.

The second portion (*b*) extends down to near Cobbannah Creek, where the nearly vertical beds, which Howitt regards as belonging to the Tabberabbera shales group, dip out of sight. Three or four claims have been worked in this area, but no particulars as to results are available.

The third portion (*c*) may be said to extend down to Lindenow Flat. Claims have been worked in various places, and gold in small quantities also obtained in some branch gullies. Here the sole formation is the Upper Devonian, whose conglomerates, shales, and sandstones form the hills through which the Mitchell river has forced its course.

Howitt visited the claim of Turnbull and Howarth, near Stony Creek. The ground washed by them is along the narrow flat bordering the river. The "wash-dirt" consists of sand, gravel, and boulders, derived from the Iguana Creek beds. The nature of the ground necessitates cradling, and the proprietors informed him that in consequence they were unable to make more than 2*l.* per man per week. Were water available for sluicing in the usual method, the yield would be more than double. The character of the gold and its fineness are given in the table; in appearance, it is indistinguishable from that found at Lower Boggy Creek or at Tabberabbera. Over the river, but slightly higher up stream, is the claim of Pope and Poore, on a point of land. The river being then up, Howitt was unable to visit it. Poore, however, informed him that the gold found was of two kinds—one of the usual scaly character, the other ragged and unworn. It is to be noted here that for many miles around there are, so far as known, no older rocks than the Iguana Creek beds.

3. Maximilian Creek.—At Maximilian Creek, the formation is essentially Upper Devonian. For, although in the upper workings at Blink Bonny Gully the underlying older nearly vertical quartzites have just been laid bare, they disappear very soon in following the creek, so that thenceforward down to Freestone Creek the only formation met with being the bed-rock, consists, with a great felstone sheet of the alternating conglomerates, sandstones, and shales, of the Upper Devonian. The gold is in character nuggety, much worn and rounded, and of a low standard. Its physical appearance would suggest that it has travelled; but it seems to Howitt utterly incompatible with the laws of gravitation that such masses of gold could have been transported from the Older Palæozoic areas by the streams which have excavated valleys such as that of Maximilian Creek. These older Palæozoic areas are to be found only at the source of Maximilian Creek, and have never yielded any gold, so far as known. Howitt can, however, conceive that all the

appearances suggestive of transport of the gold would arise from a gradual subsidence of the masses as the valleys were excavated. There are no means of estimating the vertical height above present levels at which the matrix from which the gold has been derived may have been situated.

It is said further that quartz veins bearing some gold, both in the quartz, and free in the casing and rubble, have been opened at Maximilian Creek, in Long Pat's Gully, and at the Crystal reef, Stewart's Gully. A fragment of quartz containing gold was given to Howitt as from the former spot; but he has no personal knowledge of the facts, and gives these statements to be taken for what they may be worth. It may be, however, borne in mind, that quartz veins are not at all rare passing through the sandstones and conglomerates of the Iguana Creek group.

Howitt thinks the following deductions may be made from the above statements:—(1) That gold is found in alluvial deposits which are derived from purely Devonian formations, as, for instance, the Iguana Creek beds. (2) That the gold found throughout a great part of this area possesses certain physical characters and a common ratio of gold and silver within certain bounds.

This being the case, the question may be asked, why should it be regarded as necessary in Victoria to restrict auriferous deposits to the Lower Palæozoic rocks, and to exclude the Devonian formations from the gold-bearing series? The explanation may be, that the richest deposits have certainly been met with here in Silurian areas, or in granite tracts which once were covered by strata of that age. No doubt it is the Lower Palæozoic rocks which have here, as in many other countries, been richest in gold; but facts collected from other parts of the world now show that they are not exclusively so.

There are no good grounds for assuming that gold is confined to one geological formation, or that the production of auriferous quartz veins has been limited to any period. At all periods it is probable that the causes which have produced gold-bearing veins have been more or less active, and may be still in operation under conditions which we are unable to examine. But we may observe the effects produced by such causes in more recent time than the geological eras mentioned above. For instance, iron-pyrites of auriferous character occur in the leads of Ballarat, Clunes, Daylesford, &c., which show the shapes of roots and branches of trees, and are therefore of very recent origin.

It may be that much of the gold, if not all, found in these Devonian areas of Gippsland, has been derived from the conglomerates and quartz grits which are themselves the result of the degradation of older and probably Silurian strata. Some part of the gold of the Mitchell river is

laminated, and may have been disseminated throughout the quartz conglomerates as one of their constituents, similarly to the gold which is found in spangles in the base of the conglomerates of the Carboniferous series of Nova Scotia. And it suggests itself that our Devonian conglomerates, or even the conglomerates of the Avon sandstones of Lower Carboniferous age, may be worth examination for such deposits. But another part of the gold is either ragged or else attached to small fragments of quartz. This gold would therefore seem to have been set free from veins *in situ*, and lends force to the statement that auriferous quartz has been found at Maximilian Creek.

It may hardly be necessary to point out that, in accordance with views which are now received among geologists as to the probable origin of auriferous quartz veins, there can be no sufficient reasons given why the quartz veins of the Lower or of the Upper Devonian strata should not also be auriferous. It was assumed formerly by some geologists, and the belief still lingers, that the granites and diorites have been the chief gold-producers, and that the auriferous quartz bands in the Palæozoic rocks are also the result of heat and chemical agency. But even the exponent of these views did not attribute the product of gold always to Palæozoic times. Sir Roderick Murchison states, in speaking of the Ural —“We are led to believe that in this region the noble metal, carried up by igneous rocks, was only brought together in rich veins at comparatively recent periods.”

In Victoria, the position of the gold-bearing quartz reefs of the Lower Palæozoic rocks shows that they are anterior in age to the Upper Palæozoic formations, which rest unconformably upon the strata in which they are contained, and together with which they have been denuded.

Howitt considers it undoubted that Victorian auriferous quartz reefs are due to deposition from aqueous solution. The investigation of vein-quartz by means of the microscope shows that it still contains portions of the fluid menstruum in minute cavities. Sorby has found that the fluid in the cavities of vein-quartz often contains a very considerable quantity of the chlorides of potassium and sodium, the sulphates of potash, soda, and lime, and sometimes free acids; and he clearly states, as his opinion, that every peculiarity in the structure of the quartz of veins can be most completely explained by supposing that it was deposited from water holding various salts and acids in solution. It would follow that the associated minerals, and even the gold, have the same derivation. Bischoff, in his great work on Chemical Geology, says that there seems to be an intimate connection between gold and iron, either in the state of sulphide or oxide, and likewise between quartz and gold. A silicate of gold may be prepared artificially, and it appears that, under certain

circumstances it may be dissolved in sensible amount. The silica constituting the quartz associated with gold certainly originates from the decomposition of silicates in rocks, and it may be conjectured that the gold has the same origin, possibly existing as a silicate.

The occurrence of gold as small crystals and as capillary masses is indicative of processes of reduction from compounds, and its frequent occurrence in quartz indicates the deposition of such gold-compounds from the water that deposited the quartz. The accidental discovery made by Daintree that a speck of gold, lying in a solution of chloride of gold, increased to several times its original size after a small piece of cork had by accident fallen into the solution, suggests also that gold may be distributed throughout the meteoric waters as a chloride; and an experiment made by the great chemist, Bischoff, before quoted, shows that, in adding to a solution of chloride of gold a solution of silicate of potassa, if the resulting precipitate is allowed to remain undisturbed for some months under water, a decomposition takes place, and in the silicate appear minute partly microscopical specks of gold. In remarking on this, Cosmo Newbery says that, if this is the method by which the gold reached the quartz lodes, the origin of the silica is also that of the gold. Howitt therefore feels justified in believing that the auriferous quartz reefs have been deposited from aqueous solutions permeating—say, for instance—the Silurian strata, and it follows that we may expect to find similar deposits throughout those formations which have been similarly affected. Although not directly bearing on the Devonian formations, it may not be uninteresting to follow these suggestions a step further in enquiring to what depth the auriferous reefs of Victoria may possibly be expected to descend. Very many questions would have to be considered, and the subject is obscure and difficult of reply. Assuming, however, that the depth to which the now remaining strata of Silurian age descend below any given datum-line is comparable with the height to which the granites, which have invaded and cut off that formation, rise above that same level, we may at the same time arrive at a rude estimate as to the depth to which it may be expected our auriferous quartz reefs may descend.

Looking at the Gippsland mountains for an illustration, Howitt finds that Mount Baldhead may be taken as the highest granite mass south of the Silurian area of the Crooked river country, while to the northward of the Great Dividing Range the same auriferous tract extends to Mount Buffalo, which may be regarded as the nearest highest granite mass to the north. About the Crooked river, the Upper Dargo, and the Upper Ovens, no granite is anywhere visible at the surface; but from the structure of the country, as well as on general geological considerations, it may be certainly assumed as existing beneath, at an

undefined depth. Taking the heights of Mount Baldhead and Mount Buffalo as being 4000 to 5000 ft. above the river-levels at the Crooked river, and regarding those mountains as part of the rim of the granite trough or basin including the Silurian area, he roughly assumes that the depth to which the Lower Palæozoic formations there descend below the datum-level may possibly be equal to the height of the granite above it ; or, in other words, that the denuded Silurian strata, in their folded and compressed condition, may possibly have a thickness of 4000 to 5000 ft. Such inference applied generally would lead to the belief that the auriferous quartz reefs of Victoria may extend to greater depths than human appliances can at present follow them.

The same consideration as to the origin of the auriferous veins in the Silurian formations would show that the presence of such in the Devonian rocks presents no improbability. These formations exhibit the effects of metamorphism, and of the extensive infiltration of silicious waters, although to a less extent than the Lower Palæozoic rocks.

That the Devonian formations are universally or richly auriferous is questionable ; but Howitt does not in the least doubt that many other places than those now mined will be profitably worked for gold in the Iguana Creek group ; and that for the future the Upper Devonian, or even the kindred Lower Carboniferous formations into which they appear to graduate in Gippsland, will not be entirely avoided by prospectors as being barren.

The gold of Minusinsk, Siberia, is partly ascribed to Devonian strata. Some of the New South Wales reefs, too, are in Devonian strata. The Te Anau auriferous beds of New Zealand are doubtfully referred to Upper Devonian. The Gympie and other gold-fields of Queensland are in Devonian. Some of the Cornish gold is ascribed to silicious bands in Devonian rocks.

CARBONIFEROUS.—Mention has just been made of Howitt's belief that the gold of the Upper Devonian rocks of Gippsland may be found passing also into the Lower Carboniferous strata, into which they appear to graduate.

Gold-fields have been worked in Queensland within areas where Carboniferous rocks are said to prevail.

In his report (1887) on the geological features of the Mackay District, Queensland, Robert L. Jack, the Government geologist (to whom the author is indebted for a copy of the report quoted from), makes the following remarks :—

“Coal seams of workable thickness and of good quality are known to exist in the Mackay District, and others may with confidence be expected to be discovered in localities not yet explored. To the intrusion of igneous rocks among the coal-measures, and consequent metamorphism

of the latter, the district is, however, indebted for the production of gold and other metals and minerals of economic value.

"It is now known that the Gympie Gold Field, once classed as Devonian, is nothing more than a part of the Queensland coal-measures (equivalent to a portion of the Carboniferous and a portion of the Permian formations of European geologists) which has been pierced by intrusive rocks, and to some extent mineralised. From what I have seen of that goldfield, I incline to the belief that the presence of carbonaceous rocks charged with pyrites is one of the essential conditions of the production of the gold. Daintree held that another and very important condition was the intrusion of masses of diorite.

"In several places in the immediate neighbourhood of Mackay, the doleritic or dioritic intrusive masses which interrupt the coal-measures and sometimes the coal-measures themselves, are intersected by quartz reefs. In one of these, in section 661, the country rock is a 'quartz diorite.' A reef occurs on the summit of a little hill, and underlies at 70 degrees to the south-east. A shaft cuts the reef at a depth of 6 ft., and follows it for 15 ft. on the underlie. There are on an average about 6 in. of reef, of which a-third or a-half is quartz. The latter occasionally bulges to 9 in. or thins to $\frac{3}{4}$ in. The quartz is accompanied by a clayey gangue. It occurs generally in large transparent or translucent crystals, the interstices between the crystals being filled with decomposing brown hæmatite. I was informed that two samples assayed by K. T. Staiger, of Brisbane, gave respectively 1 oz. 5 dwt. and 3 oz. 12 dwt. of gold to the ton. A number of specimens from this mine, which I saw in Mackay, contained visible gold"

According to S. Herbert Cox, the auriferous reefs and cements of the Tuapeka district, in New Zealand, are partially of Lower Carboniferous age. The oldest beds met with in the district are those which belong to the Wanaka formation of Captain Hutton. They consist chiefly of coarse-grained mica-schists, with frequent veins of quartz traversing them in all directions; and in certain places they form the country in which quartz reefs carrying gold and other minerals have been found. In the district examined, the most notable places in which gold occurs in this formation are the hills around Waipori, and again at Beaumont, on the N.E. side of the Molyneux river; but very little is known of the reefs in the latter locality. There are in connection with this formation beds of chert, &c., and these appear to be reproduced in the rocks which form the auriferous cements of the Blue Spur, and it seems probable that from them the auriferous cements just mentioned have derived their gold.

The Kakanui series, which is only, in point of fact, the higher beds of the Wanaka formation, consists of fine-grained mica-schists. But few quartz reefs have as yet been found in these beds of an auriferous

character ; but one which is worthy of mention occurred between the Blue Spur (Gabriel's Gully) and Weatherstone's. The reef was worked for a short time, and yielded payable gold ; but it was eventually cut off by a slide, and the mine was abandoned.

The auriferous cement deposits which are at present known, and of which some are worked, follow a north-westerly and south-easterly line, the Blue Spur being the farthest point to the north-west at present known. Between the Blue Spur, which is situated at the head of Gabriel's Gully, and Weatherstone's, the next deposit to the south east, is a narrow ridge of the Kakanui schists, and it is here that the quartz reef mentioned above occurred. The next deposit of importance is Waitahuna, although a few patches of cement do occur between these points, and have received a certain amount of attention from prospectors. After passing Waitahuna in a south-easterly direction, no work of importance has been done on these beds.

At the Blue Spur, the cements are enclosed in a trough, or, perhaps more correctly, a basin, as, in addition to the bed-rock on which they rest as in a trough, the schists to the north-west and south-east respectively rise to form hills of greater elevation than the cements themselves assume at the present time. The bed-rock on the north-eastern side of the basin or trough is very steep, while on the south-west side it is more shelving, the total thickness of the cements being certainly not less than 300 ft. An enormous quantity of this spur has now been removed. Formerly all the claims in this locality were sluicing claims, putting the whole of the cement through a process of ground sluicing, but this has not in every case proved remunerative : besides which it has been shown that a good deal of gold, which might have been saved, was lost in this process, and accordingly several of the claims have now erected batteries, and crush the cement, in certain cases putting everything through the stampers, and in others only certain parts of the stone which are known to be somewhat richer than the general run of the deposits.

Passing over the ridge between Gabriel's Gully and Weatherstone's, no cements are met with until after crossing the summit, but on the spurs falling into Weatherstone's, beds or benches of cement have been found, which yielded in some cases a considerable quantity of gold, while in the gullies below them no gold was to be found. This fact would point to more than one rearrangement of the beds since the gold was originally deposited with the cements. At Weatherstone's, the cements appear again, covering a considerable area of ground, but lying much lower than the Blue Spur.

The cement has been traced all through the flat towards Lawrence, and at a place called the mound a shaft has been sunk to a depth of 500 ft. through these cements ; but it has not received further attention

since the bottom was reached, so that probably the results obtained were not so satisfactory as could be wished. It is calculated that in the Weatherstone's field, $1\frac{1}{2}$ dwt. of gold per ton would pay well, which, when it is considered that at the Blue Spur $\frac{1}{2}$ dwt. to the ton is extracted, would appear probable.

At places the prospects of the wash-dirt are very rich, some returning 2 oz. 17 dwt. of gold per ton, and doubtless there is some richer even than this. It has, however, been ascertained by experience that the gold is by no means evenly distributed; but runs in patches without apparently any set rule, so that any estimate as to the richness of the deposit from mere assays is nearly, if not entirely, useless. The blue cements of Weatherstone's are capped by red ones, which, however, are only due to the oxidation of the pyrites which has taken place near the surface, as they partake of all the characteristics of the true cements, varying only in colour, and most assuredly not being a rewash of these beds. The same remarks apply equally well to the cements of the Blue Spur and Waitahuna.

The bottom on which the cements rest is very irregular, running in at least three gutters; the yield of gold also varies greatly with the bottom; and it is worthy of note that in the alluvial workings, where gold has been got at the junction of the schists and the cements, or in the passage from a true to a false bottom, known locally as the "feather edge," the yield has frequently been very high. At Waitahuna, the cements are, to all intents and purposes, the same as those before described.

The character of the cements in the Somerset claim is very different from that of those in other parts of the field, being very brown, and the schists much decomposed. At Coombe's claim, between Adams's Flat and Waitahuna, grits occur which resemble the coal-grits as developed at Pascall's (? Cretaceous). As far as can be judged, these beds are younger than the cements themselves, since the coal at Pascall's no doubt overlies the cements, and the grits rise into hills in the surrounding district (quartz hills) between Adams's Flat and the Tokomairi Plains.

From Adams's Flat, the best idea can be obtained of the flow of the glacier which deposited these beds, for from here a distinct depression in the country can be seen in the direction of Waitahuna and Weatherstone's. Moreover, the character of the beds is so identical along this line as to leave no doubt in Cox's mind that the rocks constituting the cements have travelled in this direction—namely, from north-west to south-east, and also that the means of transport has been glacier ice.

The coal formation is the one next met with in ascending order, and it is not very clear whether or no the coals of Kaitangata and Elliot

Vale, of the Mount Miscry coal-field, are of the same age or no as the lignites of Lovell's Flat and Pascall's, near Adams's Flat. Certain it is that these beds are separated by the Tokomairiro Plains, so that no absolute junction can be seen between them, nor can they be proved to be the same stratigraphically in this district.

It is true that a considerable difference exists between the conglomerates which cover the coal at Kaitangata, and the grits which overlie the lignite on the other side of the Plains; but this might be due to the lignites being the deposits on the sides of the basin, while the coals and conglomerates were deposited in the centre of the same.

There is another point to be taken into consideration—viz. that at Weatherstone's there are thin beds of coal below the cements or interbedded with them, and in the district of Lawrence there is a basin of lignite, as also at Evans's Flat, which is certainly unconformably younger than the cements. Cox leans to the opinion that the coals of the Mount Miscry field were formed during the deposition of the cements, and that the lignites are younger than and unconformable to them, and that the grits covering these have been formed by a re-wash of the conglomerates and cements.

The auriferous Maitai slates of New Zealand are referred to Lower Carboniferous.

The Rev. W. B. Clarke also observes that one of the Nelson gold-fields, in New Zealand, is along the Waimangardha river, and that river not only rises in the Carboniferous formation (which is based on granite) but runs altogether through a coal-field; and this was reported to the Nelson Government by Clouston, in September 1862. Sections across the gold-field show no interpolation of intermediate formations. Further, Gould has reported to the Tasmanian Government that he had actually found a particle of gold in a coal-seam, and this he exhibited to the Royal Society of Tasmania. Perhaps this was set free by the decomposition of bisulphuret of iron, so common in coal, and a source of gold in older rocks.

Visible gold is found in quartz pebbles in Carboniferous conglomerate in New Brunswick. Gold likewise occurs in Lower Carboniferous conglomerate at Gay's river, Nova Scotia. In New Mexico, it is met with in strata of quartzose sandstone, probably of Carboniferous age.

Some of the quartz reefs in Ladak traverse rocks of Carboniferous age. The gold of Minusinsk, Siberia, is partly ascribed to Carboniferous strata. The Tertiary alluvial gold of the Tallawang field, New South Wales, has been mostly derived from coal measure conglomerates. Traces of gold have been found in Carboniferous limestone in Somerset.

Several groups of the Gondwana system of India, which is

considered coeval with the Permian system of Europe, are believed to contain detrital gold, especially the Talchir beds, with probably Kamthis and Barakars.

JURASSIC.—The gold in the Secondary rocks of the Fitzroy Downs, and at the head of the Barcoo river, in Queensland, is referred to by Rev. W. B. Clarke as probably of Jurassic age. Belemnites and ammonites are found in the same beds with the gold-bearing quartz pebbles. Forbes, whose views concerning gold-impregnated intrusive rocks are stated in greater detail under Diorite (p. 94), and Granite (p. 97), considers the dioritic series to be nowhere earlier than the Oolitic, nor later than early Cretaceous. Deicke places the gold-veins found in chlorite-schist on the Callanda, in Graubunden, in strata of Jurassic age. He also mentions the occurrence of gold in Lias limestone at Grave, Hautes Alpes, France. The metamorphosed slates and limestones of the richest placers of Sonora, Mexico, are probably Jurassic.

TRIASSIC.—Professor Whitney has shown that much of the gold of California is contained in rocks of Triassic age. The Triassic system forms mountain groups in Sonora, Mexico, and wherever metamorphosed is auriferous.

CRETACEOUS.—The gold contained in the trachytes of the northern part of the Black Hills, Dakota, and in the Bear Lodge range, has been deposited in these rocks at the time of the intrusion, which Prof. Newton thinks was probably coeval with the elevation of the range at the close of the Cretaceous period.

Cox is uncertain whether a portion of the auriferous cements of Tuapeka county, New Zealand, described under Carboniferous (p. 78) are not of this age. The auriferous coal formation and propylite breccias are referred to Cretaceous-Tertiary.

Professor Whitney declares that the greatest amount of the gold in California belongs to the altered rocks of the Cretaceous epoch.

It is doubtful whether the gold found in Secondary rocks of the Fitzroy Downs, Queensland, belongs to Cretaceous, or Jurassic, or Triassic strata.

Griesbach (Mem. Geol. Survey India, xviii. pt. i. 45-7) determines the auriferous deposits of Afghanistan and the Hungarian Bannat to be in Cretaceous (hippuritic) limestone, which has been transformed into finely crystalline marble by the intrusion of syenitic, granitic, and trap-poid rocks, since erupted through; and metalliferous quartz veins occur along the contact-zone. The veins carry copper, lead, silver, and gold nickel and cobalt being generally also present, and sometimes even forming the leading minerals of the auriferous area.

Forbes thinks some of the dioritic auriferous rocks may be as late as early Cretaceous.

TERTIARY.—The vast gravel deposits of Tertiary age, resulting from the degradation of older strata containing auriferous veins, have formed and still form the chief depository whence man has drawn his supplies of the precious metal. Many of these Tertiary deposits have yielded no fossils which would enable their date to be more accurately referred to the several periods of the Tertiary time. But as yet, none seems to have been considered of Eocene age, and few of Miocene; and the great mass would appear to be divided between the Pliocene and Pleistocene, whose relations are hitherto but little determined in the chief localities of auriferous alluvions. In addition, not a few of the igneous rocks carrying gold or influencing its occurrence are referable to Tertiary time. The most convenient subdivision of this section, therefore, will be: Miocene and Post-Miocene.

Miocene.—About $\frac{1}{2}$ mile south-east from the gold-workings on Mayford spur, along the slope towards the Dargo, in Gippsland, Victoria, are abandoned workings, known as Synnot's claim, where several extensive excavations were made into the side of the hill, revealing a great thickness of gravel with some bands of foliated clay, in which Howitt found *Cinnamomum polymorphoides*, identified by Prof. McCoy as belonging to a fossil flora of Miocene age.

The Gippsland deposits lying between the basalt and the Silurian consist of hard silicious cement, ferruginous cement, coarse and fine gravels, clays, and occasionally impure lignites. In the report on South-Western Gippsland, it was shown that the silicious rock and cement (classed as Miocene) was older than the gravel in the lead; and the latter, on the evidence of the fossil fruit found in it, was referred to the Pliocene era, to which the leads of Ballarat are also supposed to belong. Subsequent investigations, the results of which have been published in reports on the geology of Glenmaggie, Dargo, and other parts of Gippsland, show that the silicious cement referred to is of Miocene age, and that it is somewhat older than the gravel deposits in the lead beds; it always occupies a position along the edges of the ancient channels, whose deeper portions appear to have cut through it; at the same time the difference in age is not more than the difference between the upper and lower layers of any deposit formed within a certain epoch. Both silicious rock and gravels are overlaid by basalt, which appears unquestionably to belong to the Older Volcanic period (Miocene), and this being the case, the position of the gravels as Miocene is established. In the workings of the Blue Rock (late Pioneer) claim, on the Tangil lead, where it descends below the existing river-level, are found fossil fruits, among which Baron Von Mueller identified *Spondylostrobus Smythii*, *Phymatocaryon Mackayi*, *Celyphina McCoyi*, *Conchotheca turgida*, and *Platycoila Sullivanii*—all identical with those found in the leads of the

Ballarat district. On this evidence, the Tangil lead gravel was, in the previous report, referred to the same age as the Ballarat leads, and its overlying basalt to the Newer Volcanic period. There is, however, no evidence to show that Miocene flora may not have continued to flourish into the Pliocene epoch; and though the leads of Ballarat and Tangil be of different ages, the vestiges of similar flora may be discovered in them. It is very probable that the outlines of all the main drainage courses of the Tertiary period, whether Miocene or Pliocene, were formed early in the former epoch. As regards the auriferous character of the Miocene gravels, the only place where they have been worked, or even prospected, is where a section of the old lead—cut through at either end by, and included within a large bend of, the existing river—remains about 1 mile south-east of the Tangil township. The gravels beneath the basalt were here easily detected and their quality tested. The greatest thickness of the Victorian beds of Miocene age is probably more than 600 ft.

Professor Judd concludes his remarks on the ancient volcano of Schemnitz, Hungary, by saying that the “mineral veins of Hungary and Transylvania, with their rich deposits of gold and silver, cannot be of older date than the Miocene, while some of them are certainly more recent than the Pliocene. Hence these deposits of ore must all have been formed at a later period than the clays and sands on which London stands; while in some cases they appear to be of even younger date than the gravelly beds of our Crag.”

Gold is found in the Miocene metamorphosed rocks in Costa Rica. Miocene is stated by some authorities to be the age of the mineralisation of the Ruby Hill ore deposits, Comstock lode, &c. The auriferous Ross beds of New Zealand are referred to Lower Miocene. Some of the Ballarat leads are considered to be Miocene.

Post-Miocene.—It is considered certain by others that the whole series of volcanic outbursts in the Comstock lode rocks are since the Miocene epoch.

The alluvial leads of India are found in Siwalik (Pliocene) beds.

The Desert Sandstone of Queensland has not yet afforded fossils enabling its age to be fixed. What may be its value for free gold is at present unsolved, but Daintree thinks the nature of its deposition seems to preclude the idea that the metal will be found in paying quantities, except where direct local abrasion of a rich auriferous vein-stone has furnished the supply.

The geological Survey Reports of Victoria divide the auriferous into older drifts [Middle Pliocene] and oldest drifts [Lower Pliocene]. The former are in many places covered by sheets of volcanic lava. Fossil fruits, some of a coniferous genus, allied to *Cupressinites* of Bower-

bank, are abundant in the clays overlying drifts in Victoria, New South Wales, and Queensland. These drifts are found at depths varying from 50 to 400 ft. The so-called oldest drifts of Victoria are considered undoubtedly of marine origin. Silurian bed-rock, resembling that of a rocky seashore, and masses of quartz (some 4 ft. by 5 ft.), well rounded, and large boulders, together with drift, much water-worn, constitute the auriferous stratum. The gold is in many places distributed through 20 ft. or more of drift. The observed thickness of this drift is 25 to 35 ft. where much eroded, probably reaching 100 ft. elsewhere.

Ulrich classes the Victorian gold-drifts under three heads, and describes them as follows :—

“I. The Older Pliocene or Lower Gold drift.

“The special character of this drift is that its gravelly portion is principally composed of quartz pebbles or boulders, which, as well as the gold contained in it, are perfectly rounded or water-worn. Although it is in some places—as, for instance, in the White Hills at Bendigo, the Loddon valley hills, &c.—well arranged according to size—i. e. from small to large, from the top downward—still it is more frequently observed that though the coarsest—the real boulder-drift—lies at the bottom, the superincumbent portion is variously composed in different localities. In our western gold fields—Ballarat, Castlemaine, Avoca, &c.—it consists of layers of sand and clay, of sandy and clayey, coarse and fine gravel, and where filling deep valleys and carrying the present surface drainage channels, beds of real drift-sand are not unfrequent. In the large deposits of this drift discovered on the Tangil river, Gippsland, the pebble drift, or rounded quartz-gravel, is only a few ft. thick, but covered in places by over 100 ft. in thickness of an indurated, yellow or brown, very arcaceous clay, which so closely resembles the soft, yellow Silurian sandstone—the rock-bottom of the district—that, except for its horizontal bedding, and the absence in it of small quartz-veins that are always present in the true bottom-rock, it might be easily mistaken for the latter.

“The predominating colours of this drift are either white or brown, or white and brown mottled, whilst the bottom layer or wash-dirt is often rich in a black carbonaceous clay, or shows a covering of this clay, in which remnants of trees—as branches, roots, trunks, leaves, seed-vessels, &c., all more or less carbonised, or sometimes wholly or partially converted into iron-pyrites—are often found enclosed. In some places, beds of real lignite, or brown coal, have also been found above the wash-dirt. Layers of hard ferruginous conglomerate, from a few inches to several feet in thickness, and requiring blasting operations in working, are hardly ever absent, especially near the bottom of the deposit ; and a peculiar,

extremely hard and dense silicious cement—a real quartzite—is sometimes observed in cakes near and at the top along basalt escarpments, or where the drift deposit is exposed near basalt-flows. The fact that these cement cakes, which are often many feet thick, do not extend beneath the basalt, but occur only where the drift was exposed to the atmosphere through denudation of the basalt, would tend to indicate that they are, as it were, the products of local metamorphism of sandy clay or sand, in which silicious water, produced through the denudation of the basalt in connection with atmospheric agencies, performed important parts. Touching the mode of occurrence of the drift it is twofold:—

“(a) *As Hills*, either solitary or in series, more or less connected, bounding gullies and flats, or more rarely rising in the centre of flats—as, for instance, the White Hills, Maryborough, a hill in the Loddon river flat below Guildford, &c. Where these hills are covered by basalt, they are generally of far larger size than where exposed (Loddon valley outliers, &c.), which is owing to the protection against denudation afforded by this rock to the drift: for the mode of occurrence of the latter as hills is plainly a result of denudation, the deposit having once filled old valleys in unbroken streams; but whilst first being indirectly raised through the formation of new valleys alongside, it was in aftertimes laterally cut into, and thus divided into parts (the present hills, by small gullies (the present drainage channels). The rock-bottom of these hills lies, in the higher parts of the gold-fields, generally high above the surface of the flats and gullies; but it is frequently the case that, towards the lower parts, it gradually sinks lower and lower, and not only disappears beneath the surface of the flats, but runs at last actually far lower than the rock-bottom of the latter above and below its limits, thereby proving conclusively that the old valleys had a steeper fall than the present ones. As regards the character of their mass, hills contain nearly always (and frequently strong) layers of conglomerate, whilst soft clay and sand-beds are less common.

“(b) *As so-called Deep Leads*.—In this case, the drift fills deep valleys and channels, which carry the present surface drainage, a mode of occurrence well exemplified at Ballarat, Bagshot, near Sandhurst, Eldorado, Beechworth, &c. The term ‘lead’ was originally intended only for the continuous run of paying washing-stuff, occupying mostly the deepest part of the channel, the so-called ‘gutter’; but it is now generally applied not only to the whole extent of this kind of deposit and to series of connected hills, but also to similar hills and drift channels of the Newer Pliocene, next to be considered. As will have been gleaned from the description of the hill deposits, the genuine Older Pliocene deep leads are continuations of the latter, and, to attach a stricter meaning

geologically to the term 'deep,' they ought to be considered to commence where the top of the deposit becomes level with the adjoining flats or gullies, whilst its rock-bottom runs at a lower level than that of the latter, or, where the deposit enters the flats, and becomes overlaid by more recent gold drifts. The deep leads are divided into main-trunk leads, main leads, main-branch leads, branch leads, according to their topographical relations to each other. At Ballarat, for instance, several strong basaltic flows cover extensive old watersheds, comprising a great number of branch leads, joining main-branch leads, which unite to several main leads, and these ultimately to the main-trunk leads. The line of drainage of the deep leads does not generally agree with, and is in some cases quite the reverse of, that of the present surface. A fine instance of this is observable close to Malmsbury. The old Belltopper lead runs there across the present Black Creek, and joins the old Coliban river lead close to the township, not far from the point where the old Taradale leads also join the old river lead, which latter runs from there right across the present Coliban river valley in the direction of the valley of the river Campaspe. This peculiar relation is easily understood from the geological features of the district; namely, whilst the extensive basaltic stream that covers the drift of the old valleys appears at gradually increasing heights above the present Coliban river-bed in the upper part of its course, it forms below Malmsbury the bed of the river itself, which consequently proves that the drift must lie at a considerable depth beneath the river. But as there is no evidence lower down the latter of any such deep outlet, and as, besides, the Taradale leads run southwards, whilst it runs northwards, the only conclusion we can come to is that the old-river channel—the main lead—trends eastwards towards the Green Hill, in which direction it must join the old valley of the Campaspe river.

"The discovery of the payably auriferous portions of deep leads, gutter or rise, as the case may be, is, especially in wide extensive valleys covered by basalt, a matter of great difficulty and expense, and can only be accomplished in an economic manner by systematic boring operations as already introduced at Ballarat, Beechworth, and other gold-fields. There are many places in the western gold-districts where extensive deep leads do no doubt exist, and prospecting by boring would be highly advisable. A most promising one is the large tract of basalt country bounded by the Clunes, Carisbrook, Hepburn, Glengower, &c., gold-fields, and called, beyond Joyce's Creek, the Bay of Biscay. All the old drainage channels of the area must, according to the topographical and geological features, join the old Loddon river; and as the basalt-covered valley of the latter is in the neighbourhood of Eddington hardly a mile in width, this locality is no doubt the most

favourable one for prospecting the old main-trunk lead. A bore was in fact commenced here some years ago, but the breaking of the boring-rods caused it to be abandoned before it had reached the rock-bottom.

“II. The Newer Pliocene, or Middle Gold-drift.

“This drift is generally not so extensively developed as the other gold-drifts, though it is doubtful whether what are on some gold-fields considered as the older deep leads, may not, in reality, belong to it as regards geological age, a point to be noticed farther on. It occurs, like the older gold-drift, both as hills and deep leads, and its connection with that drift is very various. For instance, where both drifts occur as hills, the older drift lies sometimes considerably higher—as, for instance, at Fryer’s Creek, Sailor’s Flat, near Vaughan, &c. ; or, as at Forest Creek and Barker’s Creek, they lie side by side, either quite separate or the younger overlapping the older ; or, as at Maldon, where the middle drift covers the older. At some gold-fields, the older drift forms hills alongside of or within flats that contain channels or leads of the middle gold-drift, covered by the upper gold-drift—for instance, at Talbot, Maryborough, Sandy Creek, near Maldon, &c. The rarest case is of both drifts occurring as deep leads in flats, the older drift resting on the bottom, being overlaid by the middle drift, and this again covered by the upper gold-drift. An instance of this exists in the lower part of Sandy Creek flat, near Newstead.

“Touching the character of its mass, the middle gold-drift differs from the older drift by the imperfectly rounded state of its quartz-pebbles and gold-particles, by a considerable admixture of Silurian rock-pebbles, and by striking colours, such as red, blue, greenish-yellow, &c., often all shades, marble-like mottled. Its mode of arrangement from top to bottom is rarely also as perfect according to size as is the case with the older drift. Where occurring as deep leads, it frequently contains layers of carbonaceous clay, full of fragments of wood and plant impressions ; yet the wood is not nearly so carbonised as that of the older drift. So-called cement or conglomerate layers occur also frequently, more especially in hill deposits ; their character is different, however, from that of the puddingstone of the older drift, being more that of a ‘breccia,’ whilst the cementing medium is hardly ever hydrous oxide of iron (brown iron-ore) in a pure state, but generally consists of a small percentage of it permeated through silicate of alumina, i. e. of indurated ferruginous clay. In some cases it is also calcareous, or consists of pure carbonate of lime, as, for instance, in some parts of the Sandy Creek lead, near Maldon.

“As regards the supposition previously advanced, that some of the

deep leads classed under the old drift might, as regards geological age, belong to this middle gold-drift, the deep lead of Epsom Flat, near Sandhurst, and some of the Ballarat leads, may be instanced. The White Hills of the former locality belong undoubtedly to the older drift consisting nearly throughout—the seventh White Hill, for example—of above 60 ft. of more or less cemented, perfectly water-worn gravel, arranged tolerably well, according to size, from fine gravel at the top to large boulders at the bottom. On the northern base of this White Hill, the Epsom lead, covered by upper gold-drift, commences ; but it contains only a few feet of cemented rounded quartz-gravel and boulder-drift at the bottom, the remainder consisting of white or variously-coloured arenaceous clay layers, which have no regular connection with the seventh White Hill gravel ; whilst, moreover, the bottom of the lead lies considerably lower than that of the White Hill. Considering these features in connection with the occurrence of small patches of genuine older drift conglomerate on the rises bounding Epsom Flat, it seems very probable that the deep lead channel is, in its upper part at least, of later formation than the hills, having been washed out in an apparently wide expanse of the older drift, and partly filled with re-washed material of the latter. At Ballarat the discovery has been made in later years that there exists a ‘high-rise’ gravel along some of the deep gutters, which latter are mostly filled with clay and drift sand, and but a few feet of boulder-wash at the bottom, evidently analogous features and relations to those just described of and between the White Hills and Epsom lead. Similar occurrences obtain probably also at other gold-fields. Touching the determination of the exact boundaries between the two drifts in cases like the Epsom Flat lead, it is quite impossible, on account of the never-absent covering of upper gold-drift and the great similarity in the material. This does not, however, affect the miner, for to him the distinction between the two drifts has little practical value.

“ III. The Post-Pliocene, or Upper Gold-drift.

“ Under this head are comprised three different deposits, viz. : (a) the Alluvial-drift, or simply ‘Alluvial,’ as the digger calls it ; (b) Recent River-drift ; and (c) Surface.

“(a) *The Alluvial drift* fills every flat and gully throughout the gold-fields, and is covered by soil and vegetation. It is generally composed of alternating layers of tough brown or bluish clays, indurated, ferruginous sands, and clayey gravel, which latter is mostly coarsest at the bottom, representing the ‘washing-stuff,’ and consists of a mixture of angular, or but very slightly rounded fragments and pebbles of quartz and all kinds of rocks, derived from the bounding rises and ranges of the

flats and gullies. Sometimes it shows only 1 or 2 ft. of tough, sandy ferruginous clay at the top, and the remainder is formed throughout of clayey gravel, which only becomes somewhat coarser towards the bottom. Again, there are gullies showing from less than 1 ft. to several feet of auriferous gravel at the bottom, covered by a far greater thickness of tough and sandy clays in alternating layers. As regards its thickness, it varies from a few feet to above 60 ft. in different localities. In the lower parts of gullies and flats it is always the thickest, thinning out towards the commencement of the gullies in the ranges. The gold it contains is generally but little water-worn; only in those gullies that separate the older or the middle drifts into series of hills occur often rich deposits of water-worn gold, in a gravel composed of more rounded quartz and rock-pebbles, evidently the remnants of the denuded older drifts. Touching the position of this drift, if in contact with the older drifts, it covers them without exception where they form deep leads and no basalt intervenes. As in cases of this kind the probability of the existence of two payable wash-dirt layers is given—namely, of one immediately resting on the older drift—the so-called ‘false bottom’; of the other lying on the true rock-bottom—careful prospecting of the upper floor is highly advisable.

“(b) *Recent River-drift*, as the name implies, consists of the occasional, generally patchy accumulations by floods of shingle, sand and clay along the courses of creeks and rivers, and is but rarely payably auriferous. It rests on the creek and river banks upon the soil covering the upper gold-drift; and, if happening to escape denudation by recurring floods, it may in course of time become also covered by a thin layer of soil and vegetation. This explains what is not unfrequently found in some river flats (Coliban, Ovens, Loddon rivers, &c.), viz. of two or more shingly or sandy beds, with thin layers of old humus or former surface soil intervening, having to be sunk through before the genuine clayey alluvial gold-drift is reached. To this recent drift belongs also the shingly material occupying the old watercourses that generally traverse most flats and wide gullies. Filling these old channels in most cases right up level with their banks, and being covered by soil and vegetation, this drift is seldom recognisable at the surface; and as it sometimes extends right down to the rock bottom, the miner working a claim on it has but a poor prospect for gold, though claims close adjoining, outside the old watercourse, may prove very rich. In deposits of this drift, broken and perfect shells of the common *Unio* are frequent, and so are trunks and other remnants of trees of the kinds lining the banks of the rivers and watercourses.

“(c) *Surface* deposit, which derives its name from its occurrence right at the top of the ground, is the most far-spread of our gold-drifts.

Being a result of the disintegration of the Silurian rocks *in situ*, and, owing to the general softness of the latter, it covers nearly every hill and range of these rocks, though being—with one exception, to be mentioned farther on—only payably auriferous in the neighbourhood of auriferous quartz reefs. Its thickness varies from a few inches to above 1 or 2 ft., according to the less or more decomposable nature of the underlying rocks; and it consists generally of clayey and sandy matter, mixed with angular fragments of quartz, sandstone, and slate, and carrying some humus supporting vegetation. The gold it contains is quite angular hackly or crystalline, and is derived from auriferous quartz reefs or leaders existing in the immediate vicinity. For this reason it affords by prospecting and washing, the best guide in the search for such reefs. Whilst the auriferous character of the Older and Newer Pliocene drifts offers only a general certainty, as it were, of the existence of gold-bearing reefs in a district, gullies filled with auriferous alluvial, especially their upper portions, afford undoubted proofs that auriferous reefs occur in the neighbouring hills and ranges; but only the prospecting of the surface on the slopes of these hills and ranges will in most cases actually lead to the discovery of the reefs themselves. The exception previously spoken of comes here, however, under consideration. It refers to auriferous surface composed of disturbed Older and Newer Pliocene drift—in fact, remnants of imperfect denudation of these deposits—always betraying its character by the very waterworn condition of the gold it contains, and generally, though not always, by smooth rounded quartz pebbles. It occurs mostly on the slopes of and in the gaps between the hills, the tops of which carry patches of the older drifts; sometimes, however, any undisturbed remains of the latter may be entirely absent in the immediate vicinity. Such surface has been worked in many parts of Maldon, Fryer's Creek, Campbell's Creek, and other districts; and the misunderstanding of its true nature has led to frequent vain attempts to discover in the neighbourhood the quartz reefs its contained gold was supposed to be derived from.

“With regard to the comparative richness in gold of the different drift deposits just described, it may be affirmed that, comparing different areas, the Older Pliocene drift is the richest; but, on taking the entire extent of each of the drifts into calculation, it seems indeed very probable that the alluvial drift, as the more extensively developed, furnishes more gold than any of the other deposits.

“The site of the richer washing-stuff in the course of the leads and alluvial gullies and flats is, according to experience, connected with certain features of the rock bottom and surrounding country. As some of the rules deducible in this respect may be mentioned the following: Rich wash-stuff occurs—(1) Where the bottom suddenly assumes a

flatter inclination or fall than it had previously. (2) Generally in the upper or commencing parts of the deposits, for the lower downward in the channels, the slower and broader the original current, the finer and more scattered the gold particles, more especially if, as it frequently happens, there are junctions of branch leads and gullies that only furnish an accession of poor or barren material. (3) At such points where the current of the water was, through some local impediment, intercepted or dammed up, suffering thereby a retardation of its velocity, and permitting in consequence a deposition of the heavier particles previously carried along—for instance both in front of and beyond narrow places in the channels, and where the latter make sharp bends. It happens, in bends especially, often that the gold, on account of the break in the current, is found thrown high up on the opposing rise, whilst in straight portions of the channels it lies more in the deepest parts or 'gutters.' Where bars or rises of the bottom rock traverse gutters, it is not unfrequently found that the richest, though generally thinnest, wash-stuff rests right on top of the bars or rises, and that the slope of the latter facing the current carries lower and poorer wash-stuff than the opposite one. (4) Rich wash-stuff may always be expected at the junctions of two or more leads or gullies, each of which is payably auriferous by itself. (5) Where auriferous quartz reefs traverse leads and alluvial gullies, either laterally or longitudinally. In the first case, the greater richness of the wash-stuff extends generally only for some limited distance below the reefs; but whilst the extent is greater if the angle of traverse is acute, the most favourable result is produced in the second case, where the reefs run actually in the channels of the drift deposits. Much depends in these respects, of course, upon the thickness and quality of the reefs, the fall of the channels, &c. (6) In the cracks and pits ('dips,' in miners' phrase) of an uneven bottom, the gold having there found a resting-place protected against the currents of the water. On this account it is also often observed that those portions of alluvial gullies and leads that run across or at right angles to the strike of the bottom rocks, especially where these are rather hard and jointed, and therefore expose the serrated edges of the beds, are richer, or contain frequently at least richer patches, than where the drift deposits run with the rocks. As a general rule, drift miners prefer, however, a soft bottom to a hard one.

"Various other conditions and circumstances connected with the deposition of rich wash-stuff might be mentioned, but a consideration of all leads to the general conclusion that such deposits are very irregular, and, both as regards superficial extent and thickness, subject to frequently very sudden changes. At one place it forms stripes, at another, irregular nests and patches, whilst its thickness may decrease from several feet to

hardly as many inches ; sometimes it is sharply defined, and abruptly terminating ; sometimes without any distinct boundary, and gradually running out all round."

Whitney's opinion of the high gravels of the Sierra Nevada of California is that they are the work of rivers of Tertiary age. That volcanic agencies have played an important part during the gravel epoch, and especially towards its close, is also perfectly clear, the heavy covering of lava over the gravels acting as a mechanical protection to that which is beneath. It may be set down too, as established beyond any possibility of doubt, that ice had nothing to do with any part of the erosion of the gravel period. Different conditions prevail in different portions of the range with reference to the development of the volcanic formation and gravel. The portion southwards from Mariposa contains hardly any rocks which can properly be called slaty, and has never been found sufficiently auriferous to make either surface or vein mining profitable. The distinguished "red" and "blue" gravels have been shown to be one, the former being merely the result of oxidation of the latter. The gold is almost exclusively limited to the quartz and metamorphic gravel. It is a general rule, with only rare exceptions, that in a given bank the richest portion of the gravel, as well as that containing the coarsest gold, lies immediately on the surface of the bed-rock. Whitney believes that this is correctly explained by the hypothesis that neither the coarsest gold nor the largest boulders were ever transported very far from the spots whence they were originally derived. On the other hand, it is an invariable rule that the coarser the gold the less is its market value, by reason of the greater proportion of silver in the native alloy.

IGNEOUS.—*Diorite*.—In the mining districts of Transylvania and Hungary occur volcanic rocks of compact porphyritic or crystalline-granular structure, and composed of plagioclase felspar, together with hornblende or with mica, and often with quartz. They have been variously described, both as to character and geological age, but are now "recognized as hornblende or mica andesites, or quartz-andesites [dacites]. They resemble in a remarkable manner the porphyrites, the diorites, and the quartz-diorites of pre-Tertiary age."

According to one authority, the Comstock lode is in andesite. Mount Davidson acted as a most powerful agent "in determining the position and character of the Comstock lode. . . . No place offers such conditions of easy fracture as the contact-plane between the ancient and deeply-bedded formations, and those later and less coherent eruptive rocks which have been superimposed upon their bases. Accordingly along this important junction occur the relics of a dyke of andesite, which seems to have been the first foreign substance to invade the

contact-plane and start the system of intruded materials which has finally resulted in the Comstock lode."

Wilkinson notices in some of the New South Wales gold-fields that hornblendic granites and intrusive greenstone or diorite are the original source whence the gold found in the alluvial deposits has been derived. At Grenfell this is very marked, a large mass of porphyrite intruding Upper Silurian schists. Quartz reefs, varying in thickness from that of a mere thread to over 10 ft., traverse the intrusive rock in a north-easterly direction, and in some instances pass into the adjoining schists; but though richly auriferous while in the former rocks, they cease to be so immediately on entering the schists.

At Gympie, D'Oyly Aplin has shown that the reefs are closely associated with greenstone dykes, and that where these do not occur, the drifts are comparatively barren.

Many of the reefs at Swift's Creek, Victoria, are situated (1) in the line of contact between intrusive masses of diorite and sedimentary rocks, and (2) in the regionally metamorphosed Silurian beyond the contact. The former penetrate through the contact into the underlying quartz-diorites; they have been found to be the richest near the surface, to dwindle away as they penetrate deeper, and to become less auriferous or cease at no great depth from the surface. It may, perhaps, be of some significance that, where the reefs discontinue, the dioritic rocks, as a rule, cease to be decomposed. Some few auriferous reefs have been discovered and worked within the diorite area; but in these cases, isolated patches of contact schists associated with them prove that the general absence of these rocks is in reality only due to denudation.

Among the modes of occurrence of gold in E. Australia, Daintree mentions pyritous diorites, as at the Gooroomjam diggings, Queensland; alluvial drifts from these contain gold in paying quantity.

The second appearance of gold in South America, according to Forbes, is totally distinct from the first (see p. 97) in mineral character, as well as in geological age, and results from the eruption of dioritic (greenstone) rocks, composed of hornblende and felspar (without quartz), which break through strata even as late as those containing Oolitic fossils, and consequently must be regarded as younger than the Oolitic period, but as far as researches have yet shown, are probably not posterior to the deposition of the Cretaceous strata. In this case, instead of quartz-veins carrying the gold from the granite into the neighbouring strata, veins of metallic sulphides and arsenides act in the same manner, and the gold is found imbedded in its metallic state in the compounds of sulphur and arsenic with iron, copper, &c.; and from some unknown cause the more superficial parts of these veins appear as a rule to be much richer in gold, which by the miners is generally supposed to decrease in depth.

The minerals commonly found in these veins are not the same as in the metallic veins mentioned as occurring with the granitic rocks under the first head, and, as far as observations have gone, the metals, tin, tellurium, tungsten, titanium, selenium, &c., are never found in the auriferous veins of later dates. Nothing could be more conclusive than the totally distinct age of these two sets of auriferous eruptive rocks, which Forbes believes to represent the only ages at which gold has been introduced into the upper crust of the globe, and thinks it probable that this generalisation may be carried into other parts of the world, if it be not altogether universal.

In his later extension of his theories to all the gold-deposits of the world, as stated under "Granite" (p. 97), Forbes deals with dioritic gold in the following terms:—

"The newer, or dioritic, outburst I have called Post-Oolitic, as the veins containing gold, and which proceed from its centres, cut through strata containing fossils of decided Post-Oolitic forms, and possibly may be as late as early Cretaceous. These strata are frequently much altered and metamorphosed by the contact of the igneous diorite, and at such points often become auriferous, or are cut by auriferous veins proceeding from the diorite head mass. Although the results of an extended examination of these deposits in Chili, Bolivia, and Peru, occupying me from 1857 to 1863, are extremely interesting, I have only had time to publish comparatively few of the observations made. Since my return to Europe, however, I have been able to collect sufficient data to show me that this occurrence of gold is not at all confined to South America, as I had at first imagined, but appears also to be common to all the other quarters of the world. I have seen auriferous diorites from Italy, and some auriferous rocks of this class are known to occur in the Ural; and, as before mentioned, I have specimens from California, and some time back received very similar specimens, through Lieut. Aytoun, from the gold districts of India; and lastly, within a very few days, I have had the opportunity of examining a fine series sent over to the Jermyn Street Museum by Mr. Aveline, the head of the Geological Survey in Victoria, which are all strikingly similar to those examined by myself in various parts of South America."

Professor Judd, when describing the ancient volcano of the district of Schemnitz, Hungary, says that "in every instance" of the centres of igneous activity in Hungary and Transylvania, "we find proofs that the more deeply seated masses of andesitic lava have, in consolidating, assumed a highly porphyritic or granitiform structure, and that the action upon these of acid gases and vapours has resulted in the decomposition of the mass, with the diffusion of valuable metallic ores throughout the substance of the rock, and their accumulation in considerable quantities

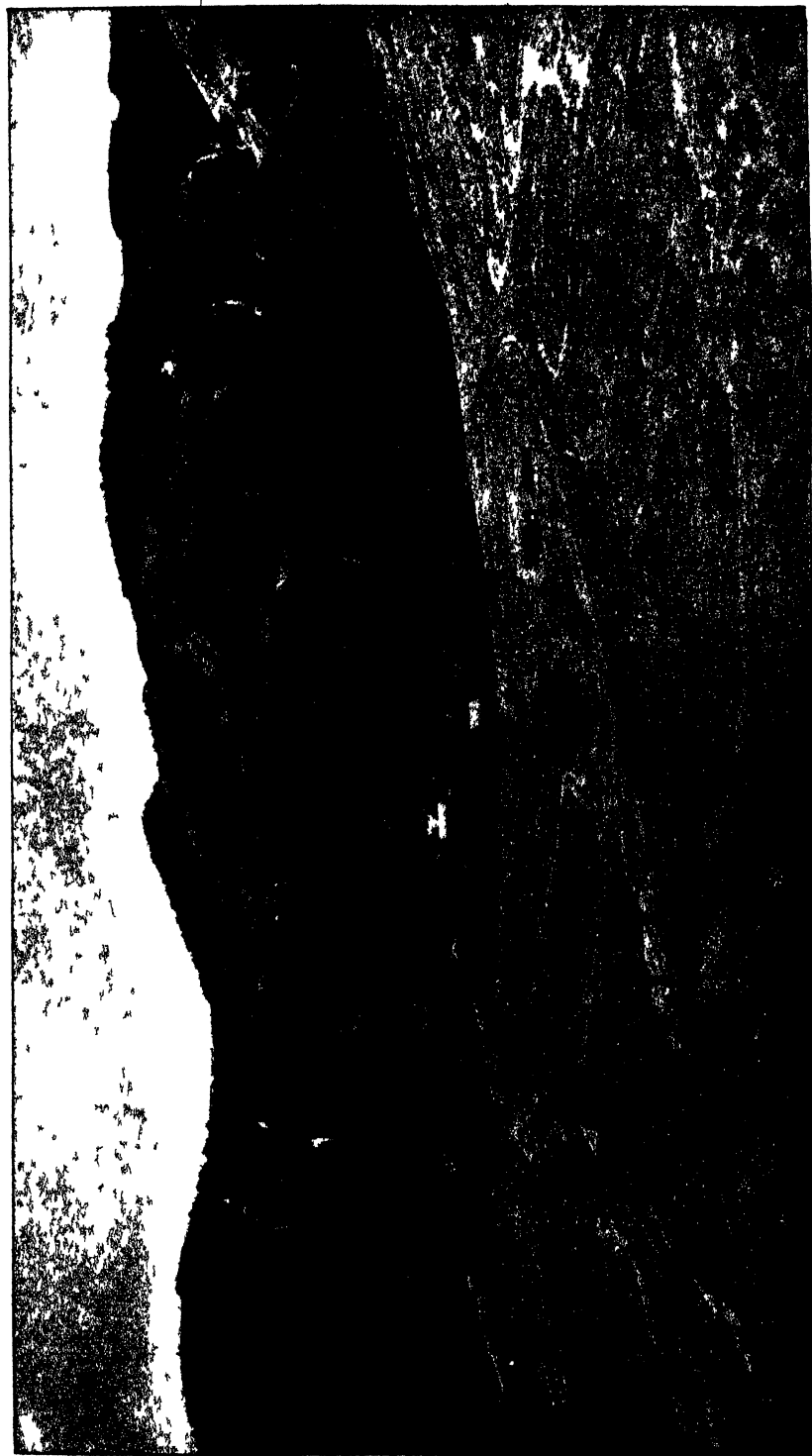
wherever a suitable fissure occurred in it." And he concludes with a remark that there is the most complete and insensible transition from the granitic rocks to the true lavas ; "and the whole of them are of Miocene date."

Diorite is the country-rock of some of the chief gold veins of Venezuela. Gold is found in the disintegrated diorites of Khutel, Turkestan. The placer-gold of the Urals, according to Murchison, lies chiefly on a greenstone bed-rock. Greenstone or diorite is the main source of much of the gold of New Guinea and New South Wales. At Temora, gold is found in the reefs traversing diorite, but not those in the slate. Some of the reefs on the Thames gold-field, New Zealand, are in diorite. Dioritic dykes accompany many of the Ballarat, Gippsland, and Western Australian veins.

Granite.—The lodes around Central City, Gilpin county, Colorado, are said to be all enclosed in rock which is of one common type, chiefly granitic, with some gneissic varieties. Gold has been found in true granite, closely resembling the variety called protogine, abundant in some parts of the Alps, at Sandy Creek, Victoria. Some of the specimens are very rich, and would assay several oz. per ton.

D'Oyly Aplin, in 1864, reported on the occurrence of auriferous quartz veins, varying in thickness from a few inches to several feet, at Wood's Point, Upper Goulbourn, Victoria, traversing in a more or less horizontal direction, and at different levels, a rock of granitoid, or rather syenitic, character, but which, existing as a broad dyke with well-defined walls, differs essentially, as a geological feature, from granite in its ordinary position as a rock mass. The rock constituting the dyke is a mixture of hornblende and felspar, with but little quartz, and occasionally mica, and might perhaps more correctly be called syenitic diorite. Crossing the dyke at different levels, some 60 or 70 ft. apart, are 3 horizontal bands or veins of quartz, more or less undulating in their course, and varying in thickness from 1 to 12 ft. As they approach the walls of the dyke, they exhibit a tendency to split up and suffer attenuation, but become exceedingly rich on entering the slates.

Interesting and apparently valuable discoveries of metalliferous lodes in granitic rocks have been made at Bethanga, east of the junction of the Mitta-Mitta with the Murray river, Victoria. The veins are of exceptional character in Victoria, inasmuch as they contain in quantity the metals, gold, silver, and copper, but they closely resemble some auriferous veins found in Queensland. The walls of the lodes are well defined. The yield of the stuff has averaged over 1 oz. per ton, the vein being 2 to 7 ft. wide. The stuff is a ferruginous gossan containing gold, silver, and copper, distributed through the ore without any apparent order.



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Morton says of the gold-bearing veins of the Virginia Gold Belt that they are generally of two classes:—(1) Veins in the slate, striking north-east and south-west; (2) veins in the syenite, striking south-east and north-west approximately, the larger veins occurring in the syenite. The characters of the veins differ in this: that in the syenite they are more persistent in length, and dip at a nearly vertical angle; while in the slate they are more irregular in length, and dip at various angles, from 25° to 80° from the horizon. The characters of the ore also differ, inasmuch as in the slate the quartz is more laminated than in the syenite, and the quantity of free gold is greater than in ores of equal assay value from the syenite.

At Kaffir's Hill, Stockyard Creek, Victoria, auriferous quartz-veins occur in a granitic dyke, and extend in the slate country on either side, which is also, together with the quartz, much impregnated by iron-pyrites.

Among the modes of occurrence of gold in Eastern Australia, Daintree specifies the granite of Bowenfels and Hartley, New South Wales.

David Forbes, F.R.S., after seven years' study of the gold deposits of South America, classes them in two categories. "Under the first head belongs all gold derived from the disintegration of granitic rocks of an age later than much, if not all, of the Silurian strata, but probably not later than the Devonian period. The largest gold-washings of South America, and probably of the whole world, I look upon as derived from this source, as well as the auriferous quartz-veins, as they may be traced to the proximity of the granite, and which I believe to have originated in or been injected from the granite into the neighbouring strata, carrying the gold, which is a normal constituent of the granite itself, along with it. This granite, wherever met with, is invariably auriferous in itself; and although it would not pay to grind down granite mountains, and work out the gold in them, yet in many parts of South America, in Brazil, near Valparaiso, &c., the granite, apparently solid, is frequently decomposed *in situ* to depths of even over 200 ft., as shown frequently in railway cuttings, and then it sometimes repays the labour of washing the whole mass for the sake of the gold in it. To this class also belong many metallic veins injected from the granite into the neighbouring Silurian strata, which contain gold, and are remarkable for the presence of other minerals, very characteristic, as oxide and sulphides of tin, tin-pyrites, copper-pyrites, compounds of bismuth, tellurium, selenium, &c., many of which are seldom or never met with in later rocks."

Subsequently Forbes extended his conclusions so far as to enunciate two epochs of auriferous impregnation throughout the whole world: (1) the older or granite outburst, and (2) the younger or diorite (see p. 94) outburst. Concerning the former he says as follows: "The

older or auriferous granite intrusion appears to have occurred at some time between the Silurian and Carboniferous periods, certainly not older than the Upper Silurian, nor younger than the Carboniferous strata, probably not younger than the deposition of the first members of the latter formation.

"Gold formations belonging to this period present themselves in Australia,* Bohemia, Bolivia,* Brazil, Buenos Ayres, Chili,* Cornwall, Ecuador, Hungary, Mexico,* New Granada, Norway, Peru,* Sweden, Ural,* Wicklow.

"To this period and cause I also attribute most of such deposits of gold as are found intruded as quartz nodules and veins in many places as if interstratified in the Cambrian and Silurian (and probably also Laurentian and Devonian) systems, which I believe to have arisen and been rendered auriferous solely from their proximity to invisible or now superficial granites."

Speaking of the celebrated Nuggety reef, Maldon, Victoria, Ulrich says, "the granite of all the veins, and that of the bottom floor, differs but slightly in universal character from the main mass of the rock in the immediate neighbourhood (it appears in the veins to be perhaps somewhat more quartzose, and to contain less mica), and there can hardly be a doubt that the second and third veins were once connected with the main mass, whilst the remainder and the bottom floor are still so, and may lead up to it if followed northward. Yet the assumption of, especially, the first 3 veins being intrusive is strangely at variance, touching their mineral connection with the reef, for there is a complete absence of any division-line between the quartz and granite—a feature generally observable between intrusive dykes and the surrounding rock. The granite appears here, in fact, not at all unlike a zone of impregnation, inasmuch as the quartz above the veins shows, first, scattered crystals and pieces of felspar; these increase gradually in quantity; plates and nests of black mica make their appearance, and, whilst their number also augments, the mixture becomes more and more fine-grained, and the passage to typical granite is insensibly completed; the reverse process of change to reef-quartz commencing again a few feet below. Even the gold takes part in the passage, for it has been found impregnated several inches deep in the granite."

At Bercosf, 10 miles north-east of Ekaterinburg, gold-quartz veins occur in dykes of very fine-grained granite, called beresite. The celebrated auriferous mispickel veins of Marmora, Ontario, Canada, are true fissures in syenitic granite, with micaceous or talcoid slates forming the walls of and horses in the veins.

* These so (*) marked, as well, I believe, as California and many others, have gold deposits of both ages.

Granite is associated with the gold-mines on Jackfish and Partridge Lakes, Canada. On ascending the Saskatchewan, the gold declines with the disappearance of granitic and gneissic rocks. Some of the auriferous veins in South Carolina are in coarse crystallised granite. In the Ili basin, Turkestan, gold is only found in the affluents which issue from syenitic granite spurs; in the waters abrading the schistose rocks, there is no gold. The Serdjiller gold mine, Asia Minor, is in syenite and mica-schist. The gold-reefs at Poverty Point, Clarence district, New South Wales, are in granitic dykes. The shallow Four-mile diggings, in the Kiandra district, have a granitic bed-rock. A granitic dyke has yielded the gold of the deep leads of the Lachlan district. The Charters Towers gold-field, Queensland, occupies the edge of an area of granite and syenite. The Black Snake gold-workings are in granite. Granite and syenite form the country-rock in the Marengo gold-field. Granite is the bed-rock of some of the Victorian leads, and country-rock of some of the veins.

Porphyry.—White states that the "Bolivia" lodes, in the Frontino and Bolivia Co.'s mines, in the United States of Colombia, are "in first-class metalliferous porphyry."

Daintree describes the occurrence of gold in porphyry at Paddy's and Sharper's Gullies, on the Cape River gold-field, Queensland. These gullies are short ravines joining the main creek about 3 miles above the junction of Running Creek and Specimen Gully. The shallow alluvial deposits in these watercourses were very rich; the material composing the drift was entirely made up of fragments of decomposed felstone-porphyry, and the rock-mass through which they had cut their channels was the solid representative of the débris in the gullies. The surface rubble on the hill-side near the head of Sharper's Gully, which is simply broken-up felstone-porphyry, was sluiced with remunerative results. The gold is of the class called by miners "dirty" or "black" gold. In the one case, it is encased in material similar to the wash-dirt from which it is separated; in the other, the coating is manganic oxide of iron, similar to the black gold of Canoona. As no portions of either of these gullies traverse the rock, and as none of the gold has quartz attached, or shows signs of having been transported from a distance, and as, whenever it is collected with adhering particles of matrix, they are invariably similar to the rock-mass which bounds the ravines, the gold has clearly been derived from the destruction of the felstone-porphyry, which here forms the bottom-rock of the miner, as its disintegrated particles make up his wash-dirt.

At another place, great care was taken in tracing the course of a peculiar porphyritic dyke, and it was found that wherever it traversed the slates on its extension, as it does where it crosses Golden Gully, near

the "Dam," there the richest gold-deposits were found. A patch of surface ground was worked at the head of a right-hand branch of Golden Gully, with "wash-dirt" and gold both similar to those in Paddy's Gully, and there can be little doubt that the red clay soil at the head of Nuggety Gully indicates the presence of this peculiar feature, and suggests the origin of its alluvial wealth. That these porphyritic elvan courses have influenced the local production of gold in the area under discussion, is further evidenced by the fact that none of the gullies between Nuggety and Specimen have been gold-producers, though the mica-slates through which they force their way differ in no respects from those worked to a profit than in the absence of these same intrusive dykes. Again, if these have not played the important part here assigned them, how shall the comparative barrenness in mineral be accounted for in the extension to the north-west of the same schistose rocks?

Porphyritic dykes are the source of the alluvial gold in Borneo. Some of the veins on the Thames gold-field, New Zealand, are in porphyry. So are a few in Queensland and in Victoria.

Serpentine.—At Native-Dog Creek, the bed-rock is Silurian shales and conglomerates, with serpentine cliffs; the shales are traversed by a dyke of quartz-porphyry. Near this dyke the rock was very rich, and gold was found in payable quantity as far as the schist and serpentine extended, but not beyond. Some of the heaviest gold was found on the schists, close to their junction with the serpentine.

At Canoona, Queensland, the gold was found to follow the course of a dyke of serpentine. Some of the Queensland reefs are in serpentine-slate. Gold occurs in the serpentine of the Quebec group, Lower Silurian, in Newfoundland.

Trachyte.—Gold is contained in the trachytes of the northern part of the Black Hills, Dakota, and in the Bear Lodge Range. No quartz whatever is associated with it. Jenney says, the gold is derived from felspar trachyte-porphyry of recent geological age.

The Report of the United States Geological Exploration of the 40th parallel places some of the prominent (auriferous) silver districts of the Carpathians, Mexico, and the United States (Comstock lode, Aurora district, Silver Mountain, Moss lode) in direct association with trachyte.

Preliminary crushings of decomposed pyritous trachytes or felsites from the Tunnel and Green's reefs, Upper Cape, Queensland, yielded about 12 dwt. of gold per ton. A closely similar property is the Peninsular reef, Portobello, Otago, where the diffused auriferous pyrites in greyish-white trachyte gave varying quantities of gold, from 3 to 11 dwt. per ton.

Von Cotta, speaking of the celebrated mine of Verespatak, the "El Dorado of Transylvania," Hungary, connects its gold-veins in some measure with the trachytic outburst of the Csétatyé.

Most of the reefs on the Otago gold-field, New Zealand, are in trachyte.

MINERAL ASSOCIATES.

The following is a brief summary of the metallic and other minerals found associated with gold and auriferous rocks, arranged alphabetically, with some remarks on the influence apparently exerted by them upon the characters of the ores.

Antimony.—Antimony sulphide (stibnite) occurs at Cata Branca, Minas Geraes, Brazil; at Paciencia and Coelho, Minas Geraes, with tellurium and iron-pyrites; with auriferous iron-pyrites and mispickel at Gold-Kronach, in the Fichtelgebirge; and in many of the Transylvanian gold-ores.

Sulphide and oxide of antimony with free gold, are found at Heathcote, Whroo, Templestowe, Caledonia, Anderson's Creek, Donovan's Creek, in the basin of the Yarra, Rutherglen, Maryborough, Blackwood, Wood's Point, Maldon, Daylesford, Ballarat, and other Victorian localities. The yield of gold from a vein of quartz and antimony sulphide at Sunbury averaged 2 oz. per ton. The antimony-mines of Costerfield give considerable quantities of gold from every vein. The ore here consists of sulphuret seamed with brown and white oxide of antimony, combined with a small proportion of auriferous quartz: below the natural water-level no oxide is found. The gold and antimony yields of some samples are thus stated:—

Locality.	Variety.	Antimony.	Gold.		
		per cent.	oz.	dwt.	gr.
Ringwood	Sulphide	33	2	5	17
Costerfield	"	45	2	15	0
"	"	42	1	19	0
Sandhurst	"	63	1	10	0
Whroo	"	65	3	18	0
Costerfield	Oxide ..	36	1	10	16
Newcastle, N.S.W. ..	Sulphide	..	0	15	0

The Costerfield sulphide on smelting gave antimony containing silver, lead, copper, bismuth, arsenic, cadmium, manganese, cobalt, nickel, chromium and iron.

In Victoria, antimony occurs native, in combination with sulphur (stibnite), as oxide (cervantite), as white antimony (valentinite), as red antimony (kermesite), and in some complex minerals, such as tetra-

hedrite and boulangerite. It accompanies gold in the veins of San Diego co., California. Some of the gold of the Argentine Republic is associated with antimony; also much of that in Bolivia. Antimony is abundantly associated with the vein-gold of Borneo, and occasionally with that of Sado, Japan. Veins of auriferous stibnite occur at Langdon's Creek, New Zealand, and in the Hohen Tauern, Bohemia.

Arsenic.—Mispickel, arsenopyrite, or arsenical iron-pyrites, is very commonly the especially auriferous ingredient of a mixed lode, e.g. the Richmond mine, Nevada.

Mispickel, having the composition of about 55 per cent. iron, 25 arsenic, and 20 sulphur, contains the greater part of the gold for which the rich and well-known veins at Marmora, Ontario, Canada, are worked. Some examples have shown very high proportions of gold.

Bismuth.—The sulphide of bismuth occurs at Cata Branca, Minas Geraes, Brazil. Metallic bismuth is found in the Victorian deep leads. Bismuthite is met with in the Victorian deep leads; and at the Chesterfield gold-district, S. Carolina.

At Nuggety reef, Maldon, Victoria, both in the neighbourhood of the granite veins (p. 98) and a few inches deep in the granite itself, occurs the rare mineral maldonite, an alloy of bismuth and gold. Hence the gold from this mine contains considerable proportions of bismuth.

Bismuth occurs with gold in Charlotte co., New Brunswick, giving 10 per cent. of bismuth and 1*l.* worth of gold per ton. The auriferous deposits of the Yeniseisk, Siberia, contain bismuth. Some of the Queensland reefs carry bismuth.

Calcium.—It has been observed by Von Cotta that lime is generally conspicuously absent from rocks associated with gold. On the other hand, in some districts of New South Wales (as Nundle and Denison, Upper Peel and Hunter rivers), the auriferous veins are composed rather of calcareous than silicious minerals, and have been yielding gold for over 20 years. The deep leads of the Lachlan, Australia, are opened on the flanks of a limestone belt; nuggets of gold weighing 2 to 9 oz. are frequently obtained from these leads, enveloped in what appears to be a decomposed silicate of lime. There was a cavern found in the limestone, 60 ft. wide and 120 ft. long, full of ordinary wash-dirt.

Two localities in the Sierra Nevada, California, are recorded as having gold in calcite or dolomite. Auriferous quartz-veins in Dog Island, Manitoba, are in dolomite. Most of the rich quartz reefs at Gympie, Queensland, contain abundance of calcite in strong veins and patches, often richly impregnated with gold. A fine specimen from these shows actual veins of largish gold specks irregularly distributed through white opaque calcite. Calcite also occurs in some of the New South Wales and Victorian reefs, and in Bohemia.

Apatite is found traversing the auriferous quartz at Pèche, Ottawa. The metamorphic Jurassic limestone of Sonora, Mexico, includes the richest gold-placers. The Atlantic Cable mine, Deer Lodge co., Montana, is in "a zone or dyke of crystalline limestone which is enclosed by granite." Both calcite and aragonite occur in some of the veins of the Thames gold-field, New Zealand. In the Queensland reef carbonate of lime, gypsum, and selenite are often met with; the Devonian reefs in Queensland are only fairly auriferous when lime or magnesia is present, and cease to be productive on entering a silicious area.

Cobalt.—Cobalt occurs with nickel in the Cretaceous gold-veins in Hungary and Afghanistan.

Copper.—Native copper in grains is found in the gravels of the Sierra Nevada, California. Copper-ores occur in the Victorian deep leads. In Colorado, very little of the gold is found free in the quartz vein-stone, but is mostly combined or intimately associated with the pyrites, the copper-pyrites being very much richer than the iron-pyrites. The copper of the Sierra Madre, Sonora, Mexico, is said to be sent to China, and to fetch a high price on account of the gold in it. In Hayti, gold is found in nests of purple sulphide of copper. Some ore of copper is associated with almost all the gold found in the Argentine Republic. At Coquimbo, Chili, gold is found in a matrix of carbonate of copper. All the gold of the Indian Archipelago contains more or less copper. The copper of Singhbhum is auriferous. Copper is associated with the vein-gold of Japan. Copper-ore and native copper occur at a placer on the Murojnaia, Siberia. The copper-ores of New Caledonia contain gold. Copper-pyrites and the carbonates occur in many Queensland reefs. Copper-ores accompany the gold in some parts of South Australia, Victoria, Bohemia, Germany, Italy, and Spain.

Diamond.—In the deep leads of Victoria, about 60 small diamonds have been found in the Beechworth district, in the ordinary wash-dirt.

In the placers of Siberia, in those along the flanks of the Appalachian chain, and in those of the Sierra Nevada, California, diamonds occur, as well as at the Cherokee mines, Butte co., California.

The frequency of diamonds in the Brazilian gold-placers is quite familiar. They occur, too, in the rivers of North-west Borneo and Matan, in the Mahanadi, India, and in Gangpur States, India.

Felspar.—Felspar, sometimes porphyritic in character, commonly forms a large portion of the gangue in the auriferous quartz-veins in Colorado.

Garnet.—Garnets are common in the Sierra Nevada gravels, California, in the Victorian deep leads, and in the South Saskatchewan gold-fields; in the Yeniseisk gold-fields; and in Hungary.

Iridium.—Iridosmine [osmiridium] occurs in the gravels of the Sierra Nevada, California, in the deep leads of Victoria, and in the gold-washings of Minas Geraes, Brazil.

Iron.—Chromite occurs in the Victorian deep leads. Iron forms an important ingredient in the celebrated *jacutinga*, the auriferous rock of Brazil. Magnetite is found in the Victorian deep leads. Limonite and earthy red hæmatite, or other ferruginous minerals, are the chief constituent of the auriferous *moco de hierro* of Venezuela. In the South Yeniseisk gold-fields, Siberia, the gold is not unfrequently covered with a thin crust of oxide of iron. Carbonate of iron occurs in some of the veins of the Thames gold-field, New Zealand, on the Normanby gold-field, Queensland, and in Victoria.

Pyrite [iron pyrites] occurs plentifully on almost all gold-fields, and is rarely free from traces of gold. At Sandhurst, Victoria, it forms the vein-stuff in some places. In California, it may be said that there is hardly any productive quartz-vein which has not some pyrites disseminated through it, yet the quantity is usually small as compared with the quartz, while in the majority of instances it is much richer in gold.

The "black" gold of Canoona, and some of that of Sharper's Gully, both in Qusensland, is coated with manganic oxide of iron. Vivianite (phosphate of iron) occurs in the Victorian leads.

Lead.—Metallic lead is very rare in gold-veins, but the ores of lead, such as galena, antimonial lead, sulphate of lead, carbonate of lead, arseniate of lead, and phosphate of lead, are often met with.

Mimetite, an arseniate of lead combined with a chloride, is found in the Richmond mine, Nevada, always rich in gold. The ore of the Richmond mine is an argentiferous carbonate of lead, with nodules of galena interspersed through it, together with a considerable amount of decomposed arsenical pyrites, which is gold-bearing. Galena is associated with gold in some of the reefs in New South Wales, with those of Otago, New Zealand, of Queensland, of Victoria, of Hungary, of Germany, of Italy, and of Scotland.

Though payably auriferous lead-mines are exceptional, no sample of ordinary lead-ore (galena) yet examined has failed to exhibit traces at least of gold. Galena is an important constituent of the gold-ores of many parts of Transylvania. The vein-gold of Arizona is almost all in argentiferous galena.

In the auriferous veins of chloro-bromide of silver at St. Arnaud, Victoria, mimetite (arsenic pentoxide 23·20, lead oxide 74·96, chlorine 2·39) appears to be pretty abundant.

The argentiferous galenas of Kulu contain gold. Galena is associated with the vein-gold of Japan. The Balgar-Dagh mines, Turkey, give ores containing 21 per cent. of lead, and 4 gr. of gold per 108 oz.

Magnesia.—Gold occurs in felsite magnesian slate in Newfoundland.

Manganese.—Some of the vein-gold in the slates of the Chaudière valley, Lower Canada, is "tarnished by a black earthy coating of oxide of manganese." Manganese is an ingredient of the auriferous Brazilian *jacutinga*. Iron manganese occurs in the Yeniseisk placers, Siberia. The "black gold" of Canoona, and some of that of Sharper's Gully, Queensland, is coated with manganic oxide of iron.

Mercury.—Rolled fragments of cinnabar occur with the stream-gold in Borneo.

Molybdenum.—Molybdenite occurs at the Excelsior gold-mine, California, and in the Hohen Tauern, Bohemia.

Nickel.—Native nickel occurs in minute rounded grains, containing traces of iron and cobalt, at Trinity Bar, 5 miles below Fort Yale, on the Fraser river. Also, with cobalt, in the Cretaceous gold-veins in Hungary and Afghanistan.

Osmium.—Laurite [sulphuret of osmium and ruthenium] occurs in the gravels of the Sierra Nevada, California. Irid-osmium occurs in the gold-washings of Minas Geraes, Brazil.

Palladium.—Palladium forms 5 to 8 per cent. of the gold-alloy in the Minas Geraes mines. Some assays made at the Rio de Janeiro mint gave the following results:—

			1.		2.		3.
Gold	..	..	88.9	..	90.25	..	92.3
Palladium	..	..	11.1	..	9.75	..	7.7

Platinum.—Platinum occurs in some abundance, with its alloys, in the gravels of the Sierra Nevada, California. The chief is native platinum, alloyed with iridium, rhodium, paladium, &c.; and platin-iridium. It is frequent with gold in the Ural placers, at Nijny Tagilsk, Bissersk, Bilimbayewsk, Bogoslofsk, Kushvinsky, Newyansk, Werh-Ysetzk, Kischtrinsk, and Miask. Also in the Cherokee mines, Butte co., California, and in the Tranquille river, British Columbia. Much of the Brazilian gold is alloyed with platinum. It is found also in Choco, United States of Colombia, in the rivers of Assam, in Mysore, and in the Rhine.

Ruby.—Rubies are found in the Victorian deep leads; some of poor quality occur in the Ayakta alluvia, Siberia.

Sapphire.—Sapphires occur in the Victorian gold-drifts: in the bed of the Tubba Rubba Creek, Mornington, and at other places, they have been found of a fine blue colour. Blue and green sapphires occur in the Mount Werong field, New South Wales.

Silica.—It is probably unnecessary to mention quartz as the most universally auriferous vein-stone; but it must not be supposed that all

quartz is auriferous, nor that all gold is quartzose. Other matrices are described in this section.

In New Zealand, "gold sometimes occurs so mixed with silicious particles as to constitute with them a golden sandstone." In Victoria, one pyritous vein consists largely of nearly pure silica, in the form of chalcedony.

A portion, at any rate, of the so-called "rusty" gold of the miners would seem to be coated with a film of some silicious mineral, probably silicate of iron, which prevents its amalgamation. Actual samples showing this feature have been gathered from a large deposit of tailings in the Feather river, below Oroville, in Butte county, California; some were wholly, others only partially coated; and while the coating in some cases was opaque, and hid the gold, in others it was semi-transparent, and revealed the metal. Similar specimens have been described by Attwood from Guayana, Venezuela.

Silver.—Silver in some form is so universally present in gold of all qualities, varying only in its proportions, that enumeration is quite impossible. It may also be said that no silver-ore is quite free from gold.

In the reefs of the St. Arnaud district, Victoria, the gold is associated mainly with chloro-bromide of silver, which has shown, on analysis, 65·14 silver, 24·16 bromine, and 10·73 chlorine; while the embolite from South American mines gives 66·562 silver, 20·088 bromine, and 13·050 chlorine.

Tellurium.—Tellurium is met with at Paciencia and Coelho, Minas Geraes, Brazil, with antimony sulphide and iron-pyrites. Tellurium is associated with some of the gold of Asia Minor.

In many of the mines of Western Transylvania, the occurrence of the gold in combination with tellurium is the common feature. Prominent examples are Nagyag, Offenbanya, &c. The chief forms are nagyagite (consisting of lead, tellurium, gold, sulphur, and small quantities of copper and silver) and sylvanite (containing tellurium, gold, and silver). These minerals are also found in the Melones and Stanislaus mines, Calaveras, California, and in the Red Cloud mine, Colorado.

Tin.—Rutile, anatase, brookite, and cassiterite occur in the Victorian deep leads. Stream-tin is found with alluvial gold in Borneo and Perak, and in the Great Penchenga valley, Siberia.

Titanium.—Menaccanite occurs in the Victorian deep leads, in the gold-sands of California, in the Yeniseisk gold-fields, and in the auriferous sands of the Dec.

Topaz.—Topazes are found in the washings from Morris Ravine, Sierra Nevada, California, and in the drifts of Victoria, some of the latter specimens being very good.

Tourmaline.—This mineral is met with in the Victorian deep leads ; in the Yeniseisk placers, notably on the Ayakta ; and in some Victorian reefs.

Tungsten.—Wolfram occurs in the Victorian deep leads, and in the Irish auriferous pyrites veins.

Scheelite [calcic tungstate] was found as a gangue for gold by Dr. C. Le Neve Foster in the Italian Alps, at the Val Toppa mine, in the Val d'Ossola, near Piedimulera, and is called by the miners *marmor rossa* ("red marble"). Prof. Silliman received a sample of gold in scheelite from the Charity mine, Idaho. Scheelite occurs in the auriferous veins of the Hohen Tauern, Bohemia.

Vanadium.—Dr. James Blake found gold associated in considerable quantity with a vanadium ore which has been named roscoelite, at the Sam Simms and Big Red Ravine mines, near Coloma, El Dorado county, California. The gold is found interstratified with laminæ of the mineral, or imbedded in it, in pieces varying from the minutest microscopic particles upwards.

Zinc.—Blende occurs in most of the gold-ores of Transylvania. It is sometimes found to be auriferous in Lower Silurian veins in Lower Canada. Auriferous blende is found in some parts of Madura, India, and in Japan. Zinc is associated with some of the gold of Asia Minor, of Otago, New Zealand, of Queensland, of Victoria, and of Italy.

Zircon.—Zircon is common, in minute crystals, in the Sierra Nevada, California ; in the Victorian deep leads ; in the Gavrilof placer, Yeniseisk, Siberia, and on the Ayakta ; in the Eucumbene river, New South Wales ; and in the Mount Werong field.

CHAPTER II.

PROSPECTING FOR GOLD-BEARING GRAVELS AND ROCKS, AND
ASSAYING AURIFEROUS MINERALS.

IN the preceding chapter have been described pretty fully the conditions under which gold has been found to occur. The next step is to consider what are the guides and indications which will assist the unscientific prospector in his search after this precious metal.

The domain of the gold-miner lies in hilly ground. Flat plains have little attraction for him, except under special conditions ; because, though valuable minerals may be present, they are certain to be covered with an enormous deposit of soil, useful only for agricultural purposes.

Outward Features.—The outward features which Australian miners look for in gold-fields are granitic, porphyritic, quartzose, or clay-slate formations, these being the most generally (though not exclusively) auriferous rocks. The first step usually taken, after roughly noting the character of the rocks occurring in the district, is to follow up the water channels. Specimens of all the rocks met with in the course of the stream are sure to be found in its bed, washed and cleaned so as to be readily recognisable.

Alluvial Gold.—Attention is primarily directed to the discovery of quartz rubble, which is very commonly stained a rusty-brown colour by peroxide of iron. When basalt or trap ranges predominate, in a country giving evidences of the breaking up of quartz reefs, by the quantity of quartz débris scattered about on the surface, valuable deposits are very likely to occur underneath the volcanic formation, as the lava streams have flowed in natural depressions, notably in the channels of rivers, which they have filled up, to the exclusion of the liquid element, hiding and preserving under a thick rock covering the gold which the stream had accumulated in the irregularities of its bed.

The existence of broken up quartz is considered a good sign, and the size of the fragments serves to indicate in a measure the proximity or remoteness of the reef whence they have been derived. But the most important evidence will be found in the fine sandy matters collected in the stream bed, whether ancient or modern, i.e., whether the river has long since ceased to exist or is running at this day. Samples of these sands should be taken, especially where eddies and back-waters have

been formed. The whole of the deposit should be scraped up, even down into every crevice and depression in the bed-rock, or solid undisturbed bottom in which the river has worn a course. The fine material thus gathered must all be subjected to the process of panning, as described on p. 154, in order to separate any gold that may be hidden among the base substances. Sometimes, when water is not easily procurable, recourse is had to vanning with a vanning shovel, a method universally adopted with the stream tin of Cornwall.

The difficulty with which any but very fine free gold can be moved by water, is a good reason for maintaining that the quality of wash-dirt must depend chiefly on the character of the local auriferous matrices. The concentrating re-distribution of old drifts has, in some cases, caused the conveyance of associated gold some distance from any apparent matrix, and much gold has evidently travelled a long way while attached to quartz or clay; but this would only partially affect the auriferous character of a drift.

Were it possible for gold to travel any distance in a horizontal direction under the ordinary denuding and eroding agencies of nature, then it would not be possible for man to retain it with the apparatus such as the sluices and boxes which he employs. Many of the operations practised for purposes of mining, are only those used on a gigantic scale by nature.

It is frequent, both in deep and shallow alluvial mining, to find the richest deposits of gold on points and slopes of the bed-rock, as well as in the deepest portions of the bed. The occurrence of nuggets on "high reef," above the level of the gutters, simply indicates that their weight enabled them to remain in their position during the deeper erosion of the neighbouring gutter, and that their original matrix is at no great distance. The inevitable conclusion is that on the character of the quartz reefs or veins in their immediate vicinity depends that of the gravels. This appears to be the true explanation of the barren quality of some gravels, such as the "non-auriferous Miocene gravels": they have simply been deposited along a line of poor quartz reefs.

The leads *following* the course of lines of auriferous quartz are more likely to maintain a continuous and equal yield than those *crossing* a number of reefs at various intervals. Gold in quartz veins frequently occurs in "shoots," alternating with barren portions; this feature also causes variations in the amount of alluvial gold.

In country where the quartz veins are small and few in number, and the shoots of gold, though rich, at wide intervals and of small surface extent, the gravel in leads, creeks, and gullies may be expected to be of corresponding character. In Upper Silurian rocks, no large belts of quartz reef have yet been proved. A few small lines of quartz reefs, traceable for comparatively short distances, or lines of dyke with

auriferous quartz veins in widely separated shoots, are the usual features of Upper Silurian quartz workings in general. Some of the shoots are of great richness, and, in some instances, are traced to considerable depths, though they only recur at wide intervals on the surface; and the alluvial workings, though exceedingly rich in patches, cease to be remunerative when followed far from the perhaps solitary reef or dyke whence the gold was derived.

An examination of the distinctive characteristics and fineness of the alluvial gold found in North Gippsland, led Howitt to entertain the belief that, within certain limits, the physical character and fineness of the gold remain constant in respect to the geological formation from which it is derived. This is summed up in the subjoined tabulated form. The examples have been selected from a considerable number, so as to compare gold from similar geological formations in localities separated as widely as possible from each other. It appears to him that the following deductions may be made:—

1. From the Lower Silurians, the atomic ratio of the gold to silver varies from 22 to 1 to 6 to 1, and the gold is nuggety, laminated, ragged, and more rarely in scales.

2. From the crystalline schists of Omeo (metamorphic), the atomic ratio varies from 4 to 1 to 1 to 1, and the gold found is laminated, ragged, and in scales.

3. From some metamorphic slates and sandstones, the atomic ratio varies from 5 to 1 to 3 to 1; and the gold found is in scales and grains.

4. The more nearly the formation approaches the crystalline schists in character, the more equal the ratio of gold to silver becomes.

TABLE SHOWING FINENESS OF ALLUVIAL GOLD FROM LOCALITIES IN N. GIPPSLAND.

Number.	Gold.	Silver.	Locality.	Geological Formation of Locality, &c.	Character of Gold.
1	64'97	34'30	Swift's Creek, Omeo ..	Metamorphic; mica-schist, gneiss, granite; pierced by greenstone and felsite dykes.	Ragged and laminated.
2	86'26	23'14	Dry Gully, Omeo ..	Ditto ditto ditto ..	Ditto ditto.
3	86'93	22'57	Big River, Omeo ..	Ditto ditto ditto ..	Ditto ditto.
4	84'96	24'23	Dargo Flat, Dargo River.	Hornblende granite, flanked by metamorphic slates and sandstones, passing on either side into Lower Silurian.	Fine, scaly, and in grains.
5	89'36	9'84	Policeman's Creek, Dargo River.	Lower Silurian	Nuggety and laminated.
6	92'05	8'62	Tucker Creek, Wentworth River.	Ditto ditto and granite ..	Fine and scaly.
7	93'97	5'62	Delegete River	Ditto ditto	Nuggety and laminated.
8	94'01	5'29	Combyingbar Creek, Bem River.	Ditto ditto adjoining a granite area.	Ragged and laminated.
9	95'11	4'56	Shady Creek, Tambo River.	Ditto ditto	Nuggety and laminated.
10	96'80	2'70	Good Luck Creek, Crooked River.	Ditto ditto	Ditto ditto.
11	97'54	2'46	Upper Boggy Creek ..	Ditto ditto	Ditto ditto.
12	94'13	5'25	Upper Boggy Creek ..	Porphyry overlaid by Upper Palaeozoic shales, sandstones, and grit.	Fine laminated.
13	94'95	4'85	Lower Boggy Creek ..	Ditto ditto	Ditto ditto
14	94'77	5'23	Lower Boggy Creek ..	Ditto ditto	Ditto ditto

Examples 4 and 5 connect both series by the physical character of the gold and by the ratio of the metals. The belt of metamorphic slates and sandstones which furnish the examples 4 and 5, flank an area of hornblendic granite, traversed by dykes of curite, greenstone, and syenite-porphry. They present in places a marked resemblance in texture to some of the fine-grained examples of gneissoid schists of the Omco district. On the other hand, No. 14 illustrates a different set of conditions, where the Lower Silurian strata are highly silicified, and are cut off by the granite without assuming any of the characteristics of the crystalline schists. In this instance, the gold belongs to the Lower Silurian series, according to its ratio, but departs from the usual character of such gold in the district by being fine-scaly.

It is, of course, not possible to point out where, underneath the overlying deposits, ancient water-courses (deep leads) may be situated. There are no sufficient data, nor can any one at present say whether they may be found conjoined in one stream, or following separate courses; neither can any one forecast whether the gold deposited in the area is much or little, whether it is concentrated in leads or widely spread in drifts. The proper practical investigation of this question is by carrying out a series of borings on a carefully considered plan.

The boundary reef of a lead may slope far more abruptly on one side than on the other, and, consequently, a bore exactly central between the two sides may bottom very shallow, though in proximity to deep ground. It is, therefore, a rare circumstance if a single bore suffices. Next in importance, however, to a knowledge of the locality and depth of deep ground, is that of the position of shallow, as in sinking for deep leads it may save many thousands of pounds to sink in the bed-rock and drive a long distance to the lead, rather than to sink on its course and contend with the water which will inevitably be most plentiful in the basalts and intervening drifts over the deep ground. The value of the information afforded by bores will consist in the knowledge of the respective positions of shallow and deep ground, and the depths to which shafts will have to be sunk before opening out and driving for the latter. As a means of testing the auriferous quality of a lead, boring alone is not to be relied on. It is essential to provide the arrangements requisite to enable the drills to be worked through drifts or strata other than hard rock.

So much excellent advice is contained in John A. Miller's "Hints to Prospectors," published in the 'Handbook of New Zealand Mines,' that, though it is primarily intended for local miners, gold prospectors in all parts of the world cannot fail to find it valuable and practical to a degree. The following abstract of Miller's paper has been condensed in parts, but not so as to omit anything of general importance.

As to prospecting in alluvial country, Miller considers the subject

under the three heads into which it naturally divides itself—namely, stream and creek prospecting, terrace prospecting, and high table-land prospecting.

“Stream and creek prospecting scarcely requires detailed discussion so far as practical miners are concerned ; but, as this volume may find its way into the hands of those who are not practical miners, but who, either by chance or from inclination, may be induced to undertake prospecting, a few of the distinguishing characteristics of this class of gold-mining will be of use.

“As a rule, those streams which cross the laminations of the reefs at right angles, or nearly so, are the richest. Gold is found very rarely in those parts of the streams where the current has been the strongest, but generally in the lee or under the shelter of projecting points of rock, where beaches are usually formed ; and wherever such beaches exist there also may gold be looked for with fair chances of success. But the straight courses of streams, especially when they cross the laminations of the bed-rock at right angles, are also often very rich in gold. As a rule, the gold in streams is deposited in the crevices of the bed-rock, which must be laid as dry as possible, and picked up to such depths as the sand descends between its laminations. Very rarely is the gold found mixed up in the gravel lying in the beds of streams.

“Terrace prospecting requires more experience, labour, perseverance, and, consequently, more time than creek prospecting. Terraces are those shelf-like excavations upon the hill-slopes flanking valleys and lakes, and are the remains of old river-beds ; so that the rules above laid down for the deposition of gold in streams apply also to terraces. Sometimes a ridge of rock divides a terrace from the newer river-channel, but this is not always the case. The first thing to do in prospecting a terrace is to discover its inlet, and next to find its outlet. This found, the “wash” should be carefully examined for the prevailing indications of gold peculiar to the locality in which the terrace occurs, and also for gold. The whole width of the channel of the terrace from the edge to the high or back reef must be carefully prospected, both along the bed-rock, and also through the whole depth of the ‘wash,’ from the surface down ; for the terrace ‘wash,’ being often deposited at different geological times, not unfrequently contains gold in layers one above the other. These terraces are very numerous in New Zealand ; and a study of the river systems of the present, as well as of the preceding geological ages, will assist the prospector greatly in his choice of a likely spot where to set in. There is a general fall in the country lying along the main directions of the present streams, which indicates the leading course upon which all alluvial gold has been deposited, the consideration of which is often a reliable guide to the level of terraces which are covered up

completely by landslips, or in which either the inlet or the outlet is hid from view. In New Zealand large fortunes lie buried in many locations thus hid, and require a great amount of prospecting to unearth; so that prospectors have an extensive field before them in this direction.

"Table-land prospecting is rather a new departure in gold-mining, and, so far as it is known, is confined to Otago. By the term 'table-land diggings' such deposits as have been found at Campbells, Mount Buster, Mount Pisa, and recently at Mount Criffel—all in Otago—are meant. In these localities exceedingly rich deposits of gold were discovered by accident. Table-land prospecting is therefore an entirely new departure for the enterprise of the prospector, for it would be nothing short of folly to expect that all the payably auriferous table-lands in New Zealand have been discovered. Against this one-sided opinion it may be advanced that the gold found at these elevations belongs to a different geological age from the gold generally obtained in New Zealand. This opinion is supported by the nature of the wash in which the table-land gold is found, nothing like it being met with on lower levels.

"In looking for new deposits of this description, the prospector should not be deterred by the elevation of any table-land, for gold in really astonishing quantities has been got at Mount Criffel, upwards of 4000 ft., and at Mount Pisa, nearly 6000 ft., above sea-level. Nor need much attention be paid to the 'wash' in these places; and, indeed, its total absence does not necessarily imply the absence of gold. As a rule, the sinking upon these table-lands is shallow and easy, and often the gold is distributed evenly throughout the body of the 'wash,' or else occurs in layers one above the other, as well as upon the bed-rock. So far as discovered, these deposits occur upon mountain-tops bordering the principal Otago goldfields, rather than upon those in their midst, the gold being found deposited in a somewhat erratic manner.

"The wealth that awaits the prospector in this direction is shown by the workings of some lucky miners at Mount Criffel, where the prospectors washed out during one summer about 1200 oz. of gold in a fortnight, and where a man and his son netted an average of 1500 oz. of gold for a season's work (about 7 months) several years in succession. This class of mining resembles what is known in California as 'pocketing,' and is in New Zealand only in its infancy, holding out, however, great chances of success to the gold-digger.

"Beach-workings may receive a passing notice, as the ocean beaches of the Middle Island have contributed considerable quantities of gold to the total yield of the colony. And the same may be said of dredging the larger rivers—an enterprise often highly remunerative. Both descriptions of gold-raising are capable of much extension and improvement."

Vein Outcrops.—Whilst working up the stream, attention must be

paid to the banks on each side, especially where they show a section of the rocks, so that no outcropping vein may be overlooked. Should a quartzose or pyritous vein be discovered, specimens must be broken down for examination.

Following Float.—As all alluvial gold is the result of the breaking up of auriferous veins, an effort should always be made to trace the former to its source. Many valuable reefs have been found in this way. The occurrence of "float," or stray pieces of quartz, if of a promising appearance, should be taken as a guide also. Obviously, some judgment is necessary in estimating the best direction for search, and in determining the drainage area to which the float belongs. Sometimes there may be much difficulty in locating the reef, owing to disturbances, or to accumulation of detritus hiding the nature of the subjacent rock. A knowledge of the neighbouring reefs and their gold is of great assistance. Where the gold specks and pebbles are much worn, they may represent a former riverine deposit, or its remains, scattered far from its previous resting-place. Where the gold is heavy and the rock fragments are angular, the reef cannot be very far distant. Occasionally there may be a distinct feature in all the veins in a district, such as a peculiar band of defined colour, and the recurrence of this indicator should then stimulate search for the accompanying reef. Coarse alluvial gold is not always incompatible with fine reef gold as a source, because the reef gold may be so fine in general as to lend itself to very wide distribution when once liberated, while the rarer coarse grains would not be transported far; further, the grains may have "grown" while lying in the gravel, as described on p. 18.

Loaming.—Much intelligent prospecting for auriferous reefs is done by "loaming," as a preliminary to cutting experimental trenches or sinking trial shafts. Loaming, long practised in searching for tin in Cornwall, consists in washing surface prospects from the bases and slopes of the ranges, until specks of gold or specimens are found to be obtainable with tolerable frequency within certain limits. The prospector then proceeds to trace the gold up-hill to its source, narrowing the limits of his work, as by slow and patient search he approaches the reef whence the gold has been derived. When he finds that he can obtain surface prospects of gold up to a certain point or line, but no farther, he then proceeds by means of trenching, &c., to search for the reef. As may be imagined, the work is one requiring patience, energy, and insensibility to fatigue, together with good memory for locality, as the prospector frequently has to work along a steep and scrubby mountain side, selecting his prospects, numbering them, and placing them in his "loam-bag," noting the localities, and then conveying his samples many hundred feet down the ranges to water. If he be fortunate in obtaining

prospects of gold, he has to find his way back to the spots the samples were taken from, so as to continue his up-hill search, and trace the gold to its matrix. In many new discoveries made by means of the above system there was no surface indication whatever of the existence of a reef; nothing visible to the eye that would have led one passing over the ground to believe that it was there, the soil and débris completely concealing its outcrop, until by means of "loaming," the prospector was enabled so nearly to ascertain its position as to expose it in a trench of not many feet in length. Loaming is in fact simply a systematic method of following float.

On the subject of vein seeking John A. Miller has some equally valuable remarks that deserve quotation. He says:—

"As a rule, all New Zealand quartz reefs run east and west, or nearly so, and in strike and dip cross the country, dipping north. When reefs conform to all these conditions they are called true reefs. The most permanent reefs in New Zealand are distinguished by these conditions. The gold in the reefs occurs in bands or shoots, from a few feet to several hundreds of feet in width, divided from each other by barren places. These shoots all dip west, but at very diverse angles, sometimes running almost flat or horizontally, sometimes dipping perpendicularly.

"The distribution of gold in quartz reefs is now beginning to be better understood. It has become a generally recognised fact that quartz reefs, as well as other metallic lodes, vary in their richness, as also in many of their leading characteristics, as they pass through different strata. The cleavage of the quartz, to which miners as a rule pay far too little attention, is freer, sharper, and better defined in the gold-bearing portions of a lode than is the case in the barren parts. Iron pyrites is generally a good indication of the presence of gold. A nice mellow clay, resembling fat to the touch, generally flanks the lode in its gold-bearing portions. These are a few general indications of gold in lodes; there are, besides, many minor evidences of its presence, which, however, vary much, even in the same claim.

"In prospecting for lodes or quartz reefs, the first places which should receive attention are the mountain-spurs; and where there are no outcrops of quartz—which are not very frequent in New Zealand, on account of the softness of the quartz—the streak-like depressions upon the faces of the spurs should be closely examined, for reefs are very often found in them. The first operation should be to wash out some of the decaying surface-rock; and, if only the trace of quartz gold is obtained, a reef is likely to exist in the neighbourhood, and trenching should at once be undertaken, and systematically followed up. The selection of the site of these trenches must be left to the intelligence and judgment of the prospector, who, moreover, must be guided by a general knowledge of the

country he is engaged upon. It may, however, be mentioned, that when reefs are found to cross each other at right angles, those running east and west are much more likely to contain gold in payable quantity than those running north and south, independent of size and other recommendations in their favour.

"The gold in the New Zealand quartz reefs is found chiefly on one wall of the reef—or on the other—seldom all through the stone. It has already been pointed out that quartz-gold occurs in shoots, with blank spaces intervening. If a quartz reef has been discovered, but no gold found in the stone, the prospector need not therefore feel disheartened, but should try the same reef in different places, and success may in the end reward his search.

"When this has happened, the prospector must guard against being over-sanguine, and ought to carefully consider the probable cost of working the mine, and the surrounding facilities. He should first take out about 50 to 60 tons, or more, from different parts of the lode, and have the parcels crushed separately, taking care not to deceive himself by sending only the best stone (judged by appearance) to the mill, and rejecting the, in his opinion, poorer stone; for it is by no means an uncommon occurrence that stone which shows little or no gold yields much better than that in which the metal is plainly visible. In any case it is his duty to thoroughly ascertain the value of his discovery before he enlists the assistance of outside capital or spends his money on machinery.

"Much as has been said upon, and much as is expected from, the encouragement of prospecting, there is a phase of the subject which is almost entirely lost sight of, and that is, prospecting in mines which have been worked and have yielded gold for a number of years and then been left, as well as prospecting in those that have been abandoned as unprofitable. No greater mistake could possibly be committed than to condemn these mines as either worked out or worthless. Evidence continues to be furnished by every goldfield of the colony that the mines referred to have not been half-prospected, and numerous cases in point may be quoted to illustrate and support this assertion. In fact, the advantages offered, in the shape of roads and tracks through many of the goldfields, should hold out inducements to prospectors which entirely new country, cut off from communication, does not afford. A systematic course of cross-driving in many of the abandoned mines might lead to better results than are to be gained by prospecting new ground.

"New Zealand miners, as a rule, are far too chary of cross-driving their quartz claims. In most other reefing countries cross-drives are put in upon a systematic plan, and these drives are then connected by rises, so as to preclude all chance of missing even the faintest leader of quartz

which, when found, is often of great value. Nor is this system so expensive as would appear upon the face of it, for the valueless stuff excavated is used for 'filling in' stopes, and thus two purposes are served by the one operation—namely, prospecting the claim and obtaining refuse for filling in. It is by no means an uncommon occurrence that where cross-driving is systematically followed, more gold is obtained from leaders outside the reef proper than the quantity yielded by the main reef. In this manner the fact is explained how gold-bearing quartz reefs have been worked for 500 and 700 years, as, for instance, in Saxony, in Germany, and in Hungary.

"An additional incentive to the prospector turning his attention to some of our local quartz-mines at present abandoned is the introduction of electricity as a motive power for mining-machinery, which cannot fail to work a complete revolution in the industry. In a great number of quartz mines at present abandoned as unprofitable, the cost of conveying the quartz to a crushing-mill was too great to afford a profit, though with facilities in this respect they would have left a fair surplus over and above working expenses. Electricity having now stood the test of severe application at the Phœnix Mine, Skippers, no further doubt need be entertained about its efficiency. An apparatus, capable of delivering 30 horse-power at a distance of 2 miles from the generators or dynamos, has been erected at Skippers, at a cost of between 4000*l.* and 5000*l.* However, this amount was greatly increased by the difficulties of transport in the absence of a passable track; and since the erection of the electric apparatus improvements have been made in the application of electricity as a motive-power, reducing the cost of the apparatus, while at the same time increasing its efficiency.

"The advantages of electricity are, that the machinery generating the motive force may be placed wherever a cheap motor, such as water-power, can be obtained. The effective power can thence be conducted to any required point, quite irrespective of levels, turns, or any other obstacle, by a wire. Next, there is the saving in working expenses, which, after the first outlay, are almost nil where water-power can be obtained to drive the generators. With such facilities many mines which would not pay working expenses a few years ago may be made to yield handsome dividends, and upon the introduction of electricity chiefly depends the development of the quartz mining industry in many parts of New Zealand.

"Another feature, peculiar to New Zealand, may be pointed out as affording a great inducement to prospectors—namely, the recent discoveries of gold in highly payable quantities in certain bands of slate-rock, divided from the rock in which they are enclosed by walls which are equally as well defined as those which enclose quartz reefs. Green

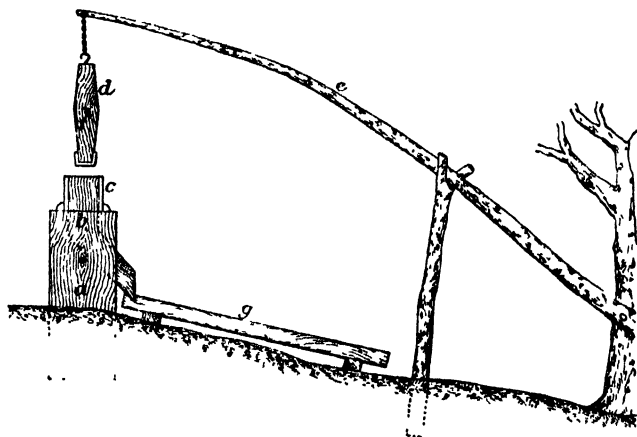
made the first discovery of this nature at the Ophir (or Black's) diggings in Otago, where, in close proximity to what has been known for many years as 'The Captain's Reef,' a great deal of prospecting was done. Green found very rich gold in the slate rock adjoining the Captain's reef. and by the aid of a common cradle washed out as much as 1 lb. of gold per week. About the same time, Southberg found gold in pure laminated slate-rock in his claim at Skippers, the British North American. In connection with this may also be mentioned the success of the Maori Point Reefing Company, which, it is reported, has obtained at the rate of 12 oz. of gold per ton from what cannot be termed anything else but a slate-reef. As in all these cases the discoveries were made by the merest accident, the attention of prospectors may well be turned to similar bands of slate rock, which are likely to reward them for their trouble."

Judging Quartz.—Though it is not always safe to trust appearances, miners often judge of the character of stone by a superficial study. Quartz which is dull white in colour, uncrystallised, and shows no traces of brownstone (decomposed pyrites) is called "hungry," and not expected to yield good returns. On the other hand, quartz which is well honey-combed from decomposition of pyrites, stained brown on the joints and faces, and shows decomposed pyrites between the crystals, promises well.

Prospecting Stamps.—In order to obtain proper samples of the vein-stuff for testing, it is necessary to reduce it to a fine state of subdivision, such as is ordinarily accomplished in the stamp battery.

A primitive yet efficient apparatus for this purpose, and such as may

FIG. 1.



DOLLY.

be erected by the prospector himself without the aid of much engineering skill, is shown in Fig. 1, and goes by the name of a "dolly." On the

end of a solid log *a*, fixed in the ground and standing about 4 ft. high, is cut a square hole *b*, about 6 in. across, in which are firmly fitted wrought-iron bars about $\frac{1}{2}$ in. apart, $\frac{1}{2}$ in. thick, and 3 in. deep, made thinner beneath, so that whatever enters above will fall through. A wooden box *c* is placed round this to keep the ore from jumping away. A square block of wood *d*, about 3 or 4 ft. long, shod with wrought iron, and small enough at the lower end to work in the box, forms the stamper. It is hung on the end of a long pole *e*, the spring of which keeps it on the swing without too much labour. It is worked by laying hold with the hands of a wooden pin *f* on each side of the stamper, and pulling it down, its own rebound and the spring of the pole taking it up again. The iron bars might be replaced by an old stamp die. The gold is caught on the table *g*.

Several forms of stamp are specially made for prospecting purposes. These will be described under "Stamp Batteries" in a later chapter.

Ore Sampling.—The following notes on ore-sampling are condensed from Dr. S. A. Reed's article in the *School of Mines Quarterly*.

The growth and multiplication (notably in Colorado) of large "custom" works for the reduction of ores of the precious metals, has caused sharp competition in the purchase of such ores; hence a prime necessity is an accurate estimation of the value of each lot which shall satisfy both purchaser and seller. It often pays better to ship ore of a certain character than to reduce it locally. The business of shipping ore has led to the establishment of a special class of mills called "sampling-works," whose sole function is to determine correctly the coin value of lots of ore brought to them. The sampler may be a special agent or buyer for some particular metallurgical works at a distance, or an independent buyer, buying all lots coming to him, and shipping on his own account to the best market for the particular kind of ore in hand, or a broker to whom the miner ships his ore to be sampled and sold by the sampler, as the miner's agent, to the highest bidder. In any case the object is to make a determination which shall be (1) accurate, (2) simple and easily inspected, and (3) cheap.

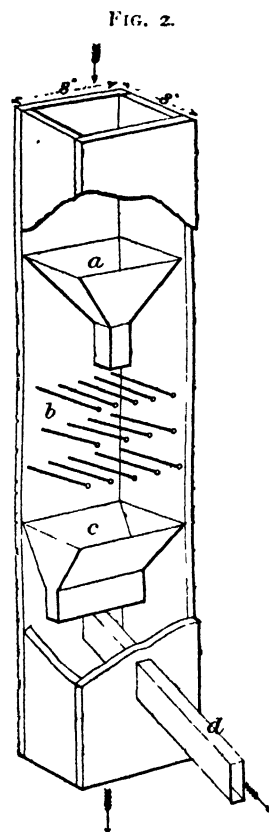
Methods of sampling vary in different works. Stamp-mills take samples from the batteries at stated intervals; some works take every tenth shovelful as a sample; others mix in a heap by raking the heap around with the shovel in a spiral, from the centre out, and then quartering, taking diagonally opposite quarters, remixing and re-quartering; others throw everything over a narrow scoop which catches a portion from each shovel; and finally, some adopt automatic devices for performing some one of these methods. The tendency is undoubtedly toward the general adoption of the scoop-shovel in some form. Automatic devices are excellent in some cases.

A Georgetown mill has a sampling device, on the intermittent principle, consisting of four cups, caused to revolve slowly under a falling stream of ore, the cups discharging their samples into a receptacle at one side.

Prof. Egleston has described several automatic sampling devices in use in Colorado.

Long experience in sampling ores in Colorado has given Dr. Reed an opportunity of testing many of the methods by comparison, and he finds the automatic arrangement about to be described, perfectly satisfactory. It is a counterpart of one previously used at a sampling-works in Clear Creek Co.

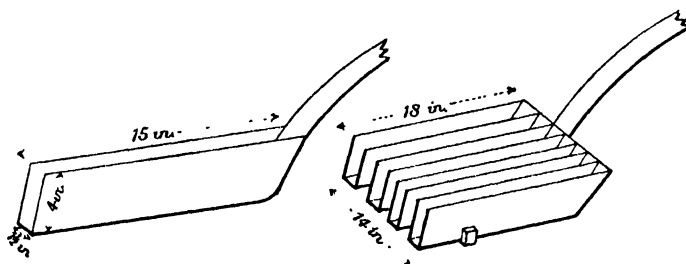
Reed passes every lot of ore, first through a crusher set tolerably fine, and then through a pair of Cornish rolls, whence it is elevated by an endless belt with buckets to a revolving drum-screen with meshes $\frac{1}{8}$ in. square, what fails to pass the screen being returned to the rolls by a chute. The ore which passes the screen goes down the chute through the automatic sampler directly into a sack held at the mouth of the chute. He uses a patent sack-fastener which can be tied up in a half a minute, and dispenses with sewing. One sack is tied while the next one is filling. The automatic sampler is shown in Fig. 2: *a* is a sheet-iron funnel by which the falling ore is thrown together into a small stream whence it falls on *b*, and is immediately scattered over the entire width of the box and falls into funnel *c*, which, from the shape of its mouth discharges a flat stream 12 in. by 1 in., and thoroughly mixed. The trough *d*, 1 in. wide, is placed directly under *c* at a sharp inclination, and carries off constantly $\frac{1}{10}$ of the stream, the remainder going directly into the sack. The trough *d* leads by a tube into a box, and the



portion conveyed there constitutes the sample. Thus when a 10-ton lot is run through the mill, a 1-ton sample is found in the box. This is then dumped on a clean floor and mixed by shovelling over a conical pile, every shovelful going to the apex of the pile. This is then thrown by shovelfuls over a scoop-shovel, constructed as in Fig. 3, or sometimes over one like Fig. 4. What is caught in the scoop, viz., about $\frac{1}{10}$ of the main sample, is dumped alternately in two heaps, making a

double sample, called original and duplicate. The two samples are then each cut down separately by throwing again over the scoop-shovel, sweeping up carefully each time, until they weigh about 10 lb. each, when they are taken to a small pulveriser, constructed like a coffee-mill, and connected with the shafting. Here they are ground to about the size of coarse sand. Each sample is then thoroughly mixed by rolling on an oil-cloth and then cut down on an ordinary tin splitter,

FIGS. 3 AND 4.



SCOOP SHOVELS.

like Fig. 4,—of rather deep and wide troughs, however—to about 2 lb., when it is ground on a rubbing-plate till all passes a 20-mesh sieve. It is then cut down on a finer splitter with $\frac{1}{2}$ -in. troughs to about $\frac{3}{4}$ lb., and this is finally ground on the plate till all passes an 80-mesh sieve.

The two pulp samples are then mixed thoroughly on glazed paper, and each divided into three portions—one for the sampler, one for the miner (who has it assayed by his own assayer as control), and one is sealed up to serve as umpire in case of disagreement; buyer and seller have each an original and duplicate, and each is assayed in triplicate, so that twelve assays are made on the lot. The triplicates should agree well, to indicate correct assay; original and duplicate should agree to indicate correct sampling; and the control must agree with the sampler, to satisfy the seller of fair dealing. A difference between sample and control of 1 per cent. on the whole value is usually settled by halving the difference, while a larger difference calls for re-assay and perhaps for re-sampling of the entire lot.

In re-sampling, the whole lot is shovelled over the scoop-shovel on the floor of the mill. When both parties are satisfied, a settlement is made at schedule rates, or the lot is put up for bids.

In some sample pulverisers the ground material falls directly over a set of segmental boxes grouped radially around the grinder. Seller and buyer then choose each a box for his sample.

Where ores are to be smelted raw, it is objectionable to have them brought to the fineness necessary for mechanical sampling. The

practice then is to pass the lot through a crusher and coarse rolls ; to sample by scoop, and pass the large samples through finer rolls called sample-rolls ; to reduce again with the scoop to a small lot which is put through a sample-grinder ; and finally to a lot which is ground fine on a bucking-plate. Automatic samplers require the material to be crushed rather fine, and to be screened, lest coarse pieces clog the apparatus.

This question of the mechanical assay of quartz was made the subject of a most interesting lecture by G. Foord before the Royal Society of Victoria, and published in their 'Transactions' (x. 139-147).

Foord remarks that there is one particular in which gold extraction differs from the treatment of the ores of most other metals ; namely, in the small proportion which the gold bears to the matrix. An iron ore may contain 30 or even 70 per cent. of metallic iron, while payable quartz may hold no more than a few dwt. of gold to the ton. If 5 dwt. per ton is the contents, the proportion in this instance is 5 times 24 = 120 gr., in the 15,680,000 gr. constituting the ton avoirdupois—about 1 part of gold in 130,000 of the ore. The gold, it is true, is in the metallic state, in isolated particles, distributed through the stone ; and the distinctive physical properties of quartz and gold are such as to be, on the whole, favourable even for the separation of so small a proportion of metal from so large a relative proportion of matrix. But oftentimes other minerals, arsenical iron, iron and copper pyrites, and many other species are present in the quartz, and it is a fact that from these natural admixtures, and from other causes, the whole of the gold is not extracted on the large scale. A proportion only is obtained from the ore, and a proportion is left in the tailings ; indeed, the technical problem is rather how to obtain the *largest profit* in working a given sample of quartz, than how to *extract, at all cost, the whole of its gold*. If the last dwt. which passes out of the mill could be extracted at an additional cost of 4s., we do better to abandon it, instead of bearing a loss which would be incidental to its recovery. The original gold contents of the ore expresses rather the extreme possibilities of its yield, than the return which we ought to expect from its treatment, but, nevertheless, there is no more valuable guide to the quartz-mill owner, or miner, than the assay, especially the assay of the tailings, collected as they pass out of the mill.

In the assay of a sample of quartz, all the gold which it contains can be separated, because there is not the same limitation of expense as that obtaining on the large scale ; we deal at most with pounds weight, and not with hundreds of tons, and the expenditure of time devoted to the work is a far more serious item than that incurred for chemicals, instruments, and fuel. With the small quantities operated on in the laboratory, means are available which would prove far too expensive for use on the grand scale.

In a large proportion of cases the quartz assay is beyond reach ; the prospector in the ranges is remote from the metallurgical laboratory, and in other cases the cost of the assay is a hindrance.

To convey a general notion of the way in which a quartz assay is usually conducted :

Suppose that specimens taken are quartz for the most part, with admixture of clay-slate, oxides of iron, and brassy iron pyrites—quite a common combination.

The gross weight of this sample is taken ; it is then pulverised finely and passed through a wire sieve of fine mesh. Of course, as gold is malleable, and as the quartz and its other accompaniments are brittle, any coarse gold existing in the sample, and separated by pounding it, may remain on the sieve, and should such make its appearance, its weight is referred to the gross weight of the stone operated on ; while for anything finer than such coarse gold, we depend on the assay of a fair sample of pulverised specimen. Weigh out a sample of the pulverised sieved stone ; in choosing the amount to operate upon, it is convenient to take a definite fraction of a ton, say a 5000th part of a ton, which is 3136 gr. Employing this quantity, it is easy to refer the result to the ton of stone, by merely multiplying the weight of gold obtained by 5000. Take this $\frac{1}{5000}$ of a ton, and roast it in a cast-iron or shallow earthenware tray, in a muffle or oven, at a dull red heat. The iron pyrites and other compounds of sulphur and arsenic, &c., in the quartz are inimical to the extraction of the gold, and one of the ways of eliminating them is this process of roasting ; burn out the sulphur and oxidise the iron with which it is combined at so low a temperature as to avoid melting any of the constituents of the sample, and this prepares for the next operation, the fusion.

To convey an idea of this fusion process (which in reality involves something more than mere fusion) consider what takes place during the manufacture of flint glass. Quartz sand forms the basis of ordinary flint glass ; during manufacture the quartz, or silica, is fused with oxide of lead and with alkali, and by the combination of these the glass is produced. It is just so in a quartz assay ; oxide of lead and alkali (carbonate of soda for instance) are added to form a fusible compound with the quartz, so indeed, that the quartz is wholly dissolved. If quartz containing gold were thus converted into a molten glass, it might be presumed that the heavy gold would gravitate through the bath of lighter molten glassy material ; and so it would if the gold happened to exist in the form of coarse grains or nuggets ; but very fine particles of gold do not very readily collect together at the bottom of such a bath, and therefore other means for the collection of the gold are involved in this fusion process.

If charcoal be heated with oxide of lead, the charcoal is consumed,

and metallic lead is revived from the oxide. Pure carbon will thus revive $34\frac{1}{2}$ times its weight of lead from the protoxide. Nitre, on the other hand, oxidises about five times its weight of metallic lead, forming oxide of lead as the result of this change. In the opposing action of these two materials, the reducing carbon and the oxidising nitre, we have the means of liberating any desired quantity of metallic lead in our bath of fused quartz, &c. However much the minerals accompanying the quartz might otherwise interfere with such a result, we can reduce metallic lead by the addition of charcoal, and we can counteract the undue reduction of lead by addition of nitre, so as to obtain a convenient quantity of reduced lead and no more.

We intimately mix our roasted quartz with oxide of lead in excess and charcoal; we add carbonate of soda, and may increase the fusibility of our bath by addition of borax or similar flux. Exposing this mixture in an earthen crucible to a red heat, the mixture melts, metallic lead is produced throughout the bath, the globules of lead pick up and alloy with the gold and gravitate to the bottom, forming a button of lead in which all the gold of the quartz is contained. The lead globules immersed in the molten bath have a *perfectly* bright metallic surface, and being under these conditions very greedy of gold, and being generated in every part of the mass during incipient fusion, no particle of gold can escape contact with and alloy with the lead. We obtain a button of auriferous lead, and a superposed layer of glassy substance or slag quite free from gold. We break the cold crucible, separate the lead button, and submit it to cupellation—that is to say, we expose this lead to a full red heat in an oven of earthenware, on a little porous cup or cupel made of the ash of burnt bones; the lead and any base metals which it may contain are again oxidised, the fusible oxide of lead with other oxides of the base metals are absorbed by the cupel, and any gold or silver derived from the sample of quartz is left as a brilliant globule on the cupel. This globule of precious metals is often very minute, and if the ascertaining the contents in gold only is our object, we may add a little pure silver to the lead on the cupel, and this will ensure a residue of convenient size; we obtain a tangible bead of silver, in which all the gold collected from the quartz by the reduced lead is concentrated. This bead, dissolved in aquafortis (nitric acid) leaves the gold as brown powder or flocks; these we can wash with two or three doses of distilled water, dry carefully, and heat to redness, whereby the full metallic aspect of gold is produced, and then the weight of this residue of gold multiplied by 5000 gives the contents of the stone expressed in terms of the ton avoirdupois.

Other modifications of the assay are practised, but the method described gives unimpeachable results; it requires a certain amount of

skill and knowledge, and therefore involves expense—expense which in many cases is prohibitive of its use.

Now to contrast this method with the mechanical assay of quartz. The ordinary treatment of quartz in the mill is itself a kind of mechanical assay; the quartz is pulverised under the stamper, in a current of water; the lighter portion—the quartz particles—are carried forward in the stream, while the gold by reason of greater specific gravity remains behind, accumulating in the ripples of the mill. The specific gravity of quartz is 2.65, that of native gold of high quality may be stated at 18.5, in other words the gold is 7 times as heavy as its own bulk of quartz. The associated minerals, the pyrites, &c., are of intermediate specific gravity, so that these are also more or less liable to be retained with the gold. There are impediments in the practice of quartz crushing, preventing a perfect extraction. The quartz is comparatively speaking only coarsely pulverised; it is never as fine as the meal of flour, for instance. Some quartz is mostly held back by the gold retained in the mill, and some gold is carried away in the escaping quartz tailings, as a result of this merely coarse pulverisation. The pyrites when separated is found to contain always some gold, at times a not insignificant portion of the whole contents; and on rewashing the tailings, fragments may be discovered, which, when examined under the microscope, are seen to consist in part of gold, in part of pyrites, and possibly also in part of quartz; and these compound particles will behave in accordance with the mean specific gravity which they possess. There is a point in the pulverisation beyond which the practical treatment of quartz on the large scale does not go, and this degree of pulverisation does not admit of a perfect mechanical separation of the gold.

But although there is this limit in the treatment on the large scale there is no practical impediment in the way of a more complete pulverisation, when, for experiment, we treat a mere handful of quartz. With a few appropriate tools and a little patience we can grind one sample until it is almost impalpably fine. We must bear in mind that the quartz, the arsenical pyrites, all the contained minerals, excepting the gold, are brittle, and that the gold alone is malleable.

The auriferous nature or otherwise of quartz may be readily ascertained by means for the most part mechanical; and we may often gain an insight into the mode of occurrence of the precious metal, and consequently as to the treatment which such a quartz would require in the mill.

The mode of procedure is simply this. First pulverise the sample in a large cast-iron mortar, with a very stout bottom; the pestle of wrought iron may be 4 ft. long, and weigh 13 lb., and the upper part of this pestle, for preservation of the hands, should be covered with a short

piece of rubber hose pipe. Employ a sieve of brass wire of fine mesh (say about 900 to the square inch), and after all the sample has passed through the sieve, grind the powder still finer in a shallow cast-iron mortar of about 11 in. diameter and 5 in. deep. Then submit a weighed quantity of the very finely ground sample to careful washing in a porcelain dish, and you will eventually obtain a residue consisting almost entirely of the metallic sulphides of the ore. These, small in bulk in proportion to that of the original quartz sample, are next transferred to a large agate mortar, again ground and washed alternately, until at last a small grey residue showing no gold is all that remains. This grey residue consists almost entirely of particles of cast and wrought iron abraded during the preceding operations. They may be easily separated. A darning needle or a steel pen is magnetised by drawing it once or twice, always in the same direction, over the surface of an ordinary horse-shoe magnet, and to the little steel magnet thus obtained, the particles of iron will attach themselves, and may be freed from any entangled particles of gold by drawing them backwards and forwards through the water with which the agate mortar is filled. The agate mortar is pre-eminently suitable for this final work; the smallest particle of gold can be seen against the dull surface of the agate; the hardest materials can be ground impalpably in the little vessel, and it serves the double capacity of mortar and wash-bowl; we can wash and grind alternately to the close of the operation.

When the gold is finally separated in spangles or flattened particles in the agate mortar, accessory chemical means for refining and collecting the precious metal into a globule may be employed. The gold residue may be wrapped in a morsel of sheet-lead and cupelled before the blow-pipe; and from the diameter of the resulting minute spherical bead of fused gold thus obtained, its weight may be easily computed. Such operations are familiar to the chemist in the laboratory, but would in most cases prove inapplicable in the hands of the miner in the field. By the purely mechanical treatment, the miner can ascertain whether the sample tried is auriferous or not, and even without attempting to weigh the separated gold, he will be able to draw valuable conclusions from its aspect in the mortar, concerning the comparative richness of the stone.

Iron Pyrites.—It is very necessary here to remind the novice that "all is not gold that glitters," and that though a good-sized lump of native gold would be pretty easily recognised, even by the uninitiated, by reason of its color, softness, and heaviness, an exceedingly small proportion is found in that state, the great bulk being mixed up more or less intimately with baser metals and minerals, so that its presence is by no means readily discoverable. Three substances which have often been

mistaken for gold are iron pyrites, copper pyrites, and mica. The two former indeed very frequently contain more or less notable percentages of the noble metal, and frequently in sufficient abundance to repay the cost of extraction. Iron pyrites or bisulphide of iron (Fe S_2), the "mundic" of the miners, is a brass-yellow coloured mineral, occurring in small cubical crystals in veins disseminated through slate rocks, and also in the coal measures, where it is known as "coal-brasses." It possesses marked characteristics which readily distinguish it from gold. When strongly heated, it is attracted by the magnet, whilst gold never becomes magnetic. Placed on a hard substance and sharply struck with the hammer, gold, being malleable, is slightly flattened without any sign of crumbling, while pyrites, being brittle, is at once reduced to powder. Gold may be cut in flakes with a pocket knife, while mundic will turn the edge of the knife and strike fire against its back, giving out sulphurous fumes. The weight or specific gravity of fine gold is about four times as great as that of iron pyrites. Gold dust is readily absorbed by mercury, while pyrites is not. Heated in nitric acid, mundic is decomposed and dissolved with effervescence and evolution of abundant red fumes, whereas gold is unaffected, and any particles of the precious metal which may have been present in the pyrites will be freed by this treatment and remain at the bottom of the vessel.

Copper Pyrites.—Copper pyrites, or yellow copper ore, the principal source of copper, is a compound of two equivalents of sulphide of iron and one of sulphide of copper. It is a deep brass-yellow coloured mineral, having a strong metallic lustre. Its primitive crystalline form is the regular tetrahedron, but it occurs in a variety of forms in lodes or veins, usually occurring in granite, grauwacke, or clay-slate. Its distinctive features are that it crumbles freely under the hammer, and yields to the knife, but instead of giving a solid chip as gold would, it produces only dust. Heated on charcoal before the blowpipe, it loses its yellow colour and fuses into a dull black globule, rendered magnetic by the iron present. Mixed with carbonate of soda and a little borax, and subjected to the blowpipe flame, it will yield (with difficulty) a button of metallic copper; but a much easier way of determining the presence of copper is to pulverise the ore, dissolve the powder in nitric acid, and evaporate the solution nearly to dryness; then add a little water, and afterwards ammonia in excess. Copper will give the liquid a rich purplish colour. When wishing to determine the presence of both iron and copper, some of the scoria from the blowpipe testing may be put into a test tube and a few drops of hydrochloric acid added to dissolve the metals. Part of the solution is then transferred to another tube, so that one may be tested for iron, the other for copper. To the first, add a few drops of ferrocyanide of potassium, and to the other, liquid ammonia.

If iron is present, the contents of the first tube will become blue, the iron combining with the ferrocyanogen and producing Prussian blue. If copper is present in the same tube, but no iron, the liquid will turn reddish-brown, owing to the formation of ferrocyanide of copper. When both are present together, their metallic precipitates will be thrown down, and a purple tint will arise from the blending of the red and blue colours. In the second tube, iron will yield a brownish tint, copper a fine blue.

To distinguish between specks of gold and pyrites in quartz, paint with a strong solution of carbonate of soda, and heat strongly in a crucible : gold melts to a yellow globule ; pyrites oxidises.

Mica.—Mica is a yellow, glistening mineral of foliated structure and semi-metallic lustre, forming one of the constituents of granite, gneiss, and mica-slate. It has frequently been mistaken for gold, though it is in reality easily distinguishable from it. Heated to redness, it becomes flakey, and loses its lustre on cooling, whereas gold would remain unchanged. But its extreme lightness should alone always suffice to distinguish it ; its specific gravity never exceeds 3·00, and it floats on mercury without uniting with it, whereas gold is immediately converted into an amalgam. Even a microscopical examination should suffice to reveal its nature.

Attwood's test.—In assaying the gold sand of the rivers, streams, and sea beaches of California, some difficulty is met with, as it contains a great amount of specular and titaniferous iron, and is called "black sand" by the miners. Platinum and iridium are often found in the same sand. Attwood, in his 'Blowpipe Assaying,' gives a convenient method of testing these sands :

"Take 100 to 1000 gr. and attack with aqua regia in a flask ; cool for about 30 minutes or more ; dilute with water, and filter. If gold is present, it will now be held in solution in the filtrate. Remove the filter, and evaporate the filtrate to dryness ; then add a little hydrochloric acid, and redissolve the dry salt in warm water ; add, to the solution so formed, protosulphate of iron, which will throw down the gold in the form of a fine, dark precipitate. This precipitate is seldom pure, being mixed with oxides of iron, and must now be dried in the filter paper, and both burned over the lamp in a porcelain dish. Then mix the dried precipitate with three times its weight of lead ; fuse, scorify, and cupel. In case platinum, iridium, &c., are found associated with the gold, an extra amount of pure silver should be added before cupellation, and the gold bottom will be found pure."

Darton's test.—Nelson H. Darton remarks that a number of methods have been proposed to detect the minute quantities of gold occurring in rocks and in ore tailings, &c. ; and having "examined and tested every

method I have known to have been suggested, I have come to the conclusion that the one noticed below yields by far the most satisfactory results if the details are well carried out, and is the most practicable in the field or in the laboratory. I know of no compound which would be formed from natural products by the method which would mislead by staining the ash a colour at all similar to the distinctive purple of finely divided gold.

" Small parts are chipped from all the sides of a mass of rock, amounting in all to about $\frac{1}{4}$ oz. This is finely powdered in a steel mortar, and well mixed. About half of it is placed in a capacious test tube, and then partly filled with a solution made by dissolving 20 gr. of iodine and 30 gr. of iodide of potassium in about $1\frac{1}{2}$ oz. water. The mixture thus formed is thoroughly agitated by shaking and warming; then, after all particles have subsided, dip a piece of pure white filter paper in it, allow it to remain for a moment, then let it drain, and dry it over the spirit lamp. It is then placed upon a piece of platina foil held in a pincers, and this heated to redness over the flame; the paper is speedily consumed; and after heating further, to burn off all carbon, it is allowed to cool, and then examined. If at all purple, gold is present in the ore, and the relative amount may be approximately deduced, as much, fair, little, or none. This method takes but little time and is very trustworthy."

Amalgamation test.—The following simple method for the detection of gold in quartz, pyrites, &c., is an adaptation of the well known amalgamation process, and serves to detect very minute traces of gold. Place the finely powdered and roasted mineral in a test tube; add water and a single drop of mercury; close the test tube with the thumb, and shake thoroughly and for some time. Decant the water, add more, and decant repeatedly, thus washing the drop of mercury until it is perfectly clean. The drop of mercury contains any gold that may have been present. It is therefore placed in a small porcelain capsule, and heated until the mercury is volatilised and the residue of gold is left in the bottom of the capsule. This residue may be tested either by dissolving in aqua regia and obtaining the purple of Cassius with protochloride of tin, or by taking up with a fragment of moist filter paper, and then fusing to a globule on charcoal in the blowpipe flame.

Panning test.—The mechanical assay of auriferous sands or stamped ore, based on common pan-washing, is of the utmost practical value to the miner, and is thus minutely described in a report of the State Mineralogist of California:

"It must be understood that this is only a working test. It does not give all the gold in the rock, as shown by a careful fire essay, but what is of equal importance to the mine owner, mill man, and practical miner, it gives what he can reasonably expect to save in a good quartz

mill. It is really milling on a small scale. It is generally very correct and reliable, if a quantity of material be sampled. The only operation which requires much skill is the washing, generally well understood by those who are most likely to avail themselves of the instruction. These rules apply equally to placer gravels. Take a quantity of the ore—the larger the better—and spall it into pieces of less size than an egg. Spread on a good floor and, with a shovel, mix very thoroughly; then shovel into three piles, placing one shovelful upon each in succession, until all is disposed of. Two of the piles may then be put into bags. The remaining pile is spread out on the floor, mixed as before, and shovelled in the same manner into three piles. This is repeated according to the quantity sampled, until the last pile does not contain more than 30 lb. of ore. As the quantity on the floor becomes smaller, the lumps must be broken finer until the last, when they should not exceed 1 in. in diameter. What remains is removed to an iron slab and, by the aid of an iron ring and hammer, reduced to the size of peas. The whole 30 lb. is then spread out, and after careful mixing, portions are lifted with a flat knife—taking up the fine dust with the larger fragments—until about 10 lb. have been gathered. This quantity is then ground down fine with the muller, and passed through a 40 mesh sieve. If the rock is rich, the last portion will be found to contain some free gold in flattened discs, which will not pass the sieve. These must be placed with the pulverised ore, and the whole thoroughly mixed, if the quantity is small; but if large, must be treated separately, and the amount of gold calculated into the whole 10 lb., and noted when the final calculation is made.

“From the thoroughly mixed sample, 2 kilogrammes (2000 grammes) must be carefully weighed out. This is placed in a pan, or, better, in a batea, and carefully washed down until the gold begins to appear. Clean water is then used, and when the pan and the small residue are clean, most of the water is poured off and a globule of pure mercury (which must be free from gold) is dropped in—a piece of cyanide of potassium is also placed with it. As the cyanide begins to dissolve, a rotary motion is imparted to the dish—best done by holding the arms stiff and moving the body. As the mercury rolls over and ploughs through the sand, under the influence of the cyanide, it will collect together all the particles of free gold. When it is certain that all is collected, the mercury may be carefully transferred to a small porcelain cup, or test tube, and boiled with strong nitric acid, which must be pure. When the mercury is all dissolved, the acid is poured off, more nitric acid is applied cold, and rejected, and the gold is then washed with distilled water and dried.

“The object of washing with acid the second time is to remove any

nitrate of mercury which might remain with the gold, and which is immediately precipitated if water is first used.

"The resulting gold is not pure, but has the composition of the natural alloy. Before accurate value calculations can be made, it will be necessary to render the gold pure and weigh it carefully.

"To purify the gold, it must be melted with silver, rolled out, or hammered thin, boiled twice with nitric acid, washed, dried, and heated to redness.

"The method of calculating this assay is very simple. It will be observed that 2000 grammes were weighed out. Let the 2000 grammes represent a ton of 2000 lb., then each gramme will be equivalent to 1 lb. avoirdupois, or one 2000th part of the whole, and the decimals of a gramme the decimals of a pound. Suppose the ore yielded, by the assay just described, fine gold weighing .072 gramme, it must be quite evident that a ton of the ore would yield the same decimal of 1 lb. Now, as 1 lb. of gold is worth \$301 46, it is only necessary to multiply this value by the weight of gold obtained in grammes and decimals to find the value of the gold in a ton of ore— $\$301\ 46 \times .072 = \$21\ 70$.

"Care must be taken in this assay to keep the cyanide solution rather weak, as gold is somewhat soluble in strong solution of cyanide of potassium, and to remember that cyanide is deadly poison which should be handled with great care."

Touchstone and Needles test.—"Select from the samples several pieces, to represent as fair an average as possible, and divide each of them with the cold-chisel. Then with each piece, using the fresh cut edges, make parallel marks on the touchstone, and lay the pieces of gold on the table in the same succession. Wet the gold streaks on the stone with nitric acid, using a glass rod or the stopper of a coin test. If no reaction takes place, and the streaks look as bright and metallic as before, the gold is at least 640 fine, and probably finer even than that; wipe the stone gently with a piece of soft rag, and apply test acid in the same manner; if there is still no reaction, the gold is finer than 750; if any action is observed, the fineness is between the two. Test acid is made by mixing 98 parts of pure nitric acid of 37° Beaumé with 2 parts of hydrochloric acid of 21° B., and 25 parts of distilled water by measure. If the golden streaks are not acted on by nitric acid, nor by the test acid, take a touch needle marked 700, and make a similar streak on the stone below that made with the samples. Compare the colour, and then progress with other needles, both copper and silver, using a higher mark each time, until a colour corresponding to that of the samples; an approximate knowledge of the quality of the gold will thus be obtained.

"But, should nitric acid cause any change in the appearance of the streaks on the touchstone, and the preliminary test in the watch glass

indicated copper, try the copper needles and apply them in the reverse order until you hit the colour, and find a needle, the streak of which is acted upon in a similar manner by nitric acid. If silver was indicated, use the silver needles. Considerable practice and a good eye are required to obtain accurate results with the touchstone, but this is soon acquired.

"Gold dust and retorted amalgam should be examined for mercury. This is done by putting a small fragment into a glass tube, closed at one end, observing that it falls quite to the bottom. Place the end of the finger loosely over the opening, and heat the closed end of the tube where the piece of gold lies, in the flame of the spirit lamp. If mercury is present, a bright ring will form in the tube above the assay. Upon examination with a magnifying glass, the ring will be found to be minute globules of mercury. To be certain, make a scratch with a file below the ring, and break off the closed end of the tube. Place the end of the now open tube into a few drops of water in a watch glass, and then, with a feather, or small stick, the sublimate may be brushed into the water, and by gently shaking, be caused to coalesce into a single globule in which form it cannot be mistaken for any other substance." (State Mineralogist of California.)

Blowpipe Assay.—The above quoted State Mineralogist says that a person "skilled in the use of the blowpipe, possessing a good balance, can make perfectly accurate assays of bullion or gold dust, but the results will only approximate, unless the whole lot is melted into a bar; as this, however, is not always convenient, the following plan may be adopted:

"Pour the gold dust out on a large and perfectly clean sheet of paper, and with the ends of the fingers, mix it thoroughly, occasionally lifting the edge of the paper to throw it together, and again mixing to ensure uniformity; then, from various parts, lift small portions, until more than 1 oz. is collected—this is best done with a flat knife, or by pinching with the thumb and forefinger; from this, weigh out accurately 1 oz. troy; place this in a small crucible, add a little borax, carbonate of soda, and nitrate of potash, and melt the whole together—this may easily be done in a blacksmith's forge or in a coal stove; when perfectly melted, set the crucible aside, and when cold, break and remove the gold button; this must be freed from clay and slag by light blows of the hammer on its edges, and subsequent washing. When perfectly clean and dry, weigh again. The loss is water, iron, sand, mercury, and other impurities, which may be assumed to be the average of the entire lot. Cut off a small portion from each side with a cold-chisel, wrap the pieces in paper to prevent them from flying, and hammer down on the anvil until thin enough to cut with scissors; place upon charcoal, and heat with the blowpipe flame until the paper is burned away, taking care not

to melt the gold. Cut with a pair of shears sufficient to weigh exactly $\frac{1}{10}$ gramme, or 100 milligrammes; a portion should be taken from each of the pieces. The weighing must be conducted with the greatest accuracy, for the success of the assay depends upon precise manipulation.

"Blowpipe cupels are made of the finest washed bone ash, formed in a cupel mould of boxwood or ebony, hammered with sufficient force to make them compact, and well dried. They are about $\frac{1}{2}$ in. in diameter, and less than that in height. For convenience, they may be supported in a ring of platinum wire, in a handle of cork, or fused into a glass rod, as described in works on the blowpipe.

"The assay is continued as follows: Place the assay in the centre of a piece of lead foil about $\frac{1}{2}$ in. square; fold the lead over the gold, and with the fingers carefully form it into a ball and set it aside. Prepare two assays like this. Take the cupel support in the left hand, and, having lighted a spirit lamp, lift one of the cupels by placing the end of the forefinger in the concave part, and holding it lightly with the thumb, place it in the loop of wire. Heat the cupel by urging the whole of the flame upon it, producing in doing so a roaring sound. This is best done by holding the point of the blowpipe outside the flame. When the cupel is hot enough, which is known by its becoming white after first blackening, lift with the pliers one of the assays, and place it in the centre of the cupel. A steadily-pointed blue flame must then be directed upon the assay until it melts and begins to oxidise, when the flame is changed to a roaring blast, and the cupel is moved farther from the lamp. Cupellation goes on rapidly if the flame is directed against the cupel beyond the assay, and not directly upon it, and if the cupel is kept cool—that is to say, at the lowest temperature at which the lead can be kept fluid. It will be found advantageous to discontinue the flame for an instant occasionally, and to direct it by short puffs at times. The exact point can only be attained by removing the cupel from the lamp, and returning it gradually, as may be required. As the cupellation goes on, the bead becomes more spherical; little patches of lead oxide form and pass to the cupel, becoming thinner, until at last the gold bead can be seen through the slight film of oxide. When nearly finished, the molten gold spits up towards the flame. At last, at the proper moment, learned only by practice, an instant cessation of the blast causes a flash, and a bright yellow golden bead remains on the cupel. When cold, the bead is removed from the cupel with pliers, and placed flat side down on a clean piece of paper. It is then grasped with a large pair of pincers and squeezed by a strong pressure. This generally removes all adhering bone ash, and renders the button fit for weighing. To make sure, turn it over, examine with the magnifying glass, and brush with a small short-bristle

brush. If anything should be found attached to it, a squeeze at right angles with the first will generally remove it. Place the button in the pan of the balance and weigh it carefully. Its weight in milligrammes is the total fineness in hundredths. For instance, 74 milligrammes would be 740 fine. With a delicate balance, thousandths can be weighed, each tenth of a milligramme being '001.

"The button will probably contain silver. To ascertain the fineness of gold, it must be subjected to a second process. The weight of the bead being noted, a cavity is made in a piece of charcoal, held by means of a proper support. In the cavity is placed the gold button, with four or five times its volume of pure silver, and both metals are melted together, before a strong blow-pipe flame. The alloy must be thoroughly fused. When cool, it is wrapped in paper, hammered flat, heated red hot, to burn away the paper, cleaned with the stiff brush, placed in a test tube with nitric acid, and boiled over a spirit lamp until no more red fumes are given off. A black powder, which is gold, will remain. The tube is then filled up with distilled water, which is poured off carefully, so as not to permit any of the finely divided gold to pass away with it. This must be repeated, and the tube filled full for the third time with distilled water. A porcelain cup is then placed over the tube, like a cap, and both inverted together. The gold falls to the bottom of the cup, and the tube is carefully removed. The water is then poured from the gold in the cup, which is first subjected to a gentle heat, and then made red-hot by the aid of the blowpipe. During the process, the cup may be held, by the aid of pincers, over the flame of the spirit lamp, which is urged upward against it from below. When the gold has assumed its metallic colour, the operation is finished. When cold, the gold is brushed into the pan of the balance, and its weight in tenths of a milligramme, noted. The results may be written as follows: Suppose the weight of the cupelled button, in milligrammes, to be 74.4, the total fineness will be 744; weight of gold powder, $69.2 = 692$; fineness of silver, 52. Or, fineness of gold, 692; fineness of silver, 52; total fineness, 744.

"It is important in estimating the value of purchased gold dust, to carefully examine, to see if there is any counterfeit, or, as it is called, "bogus" dust present. If all from the same locality, the dust will have a uniform color. Any suspicious looking pieces should be set aside and cut with a cold chisel while lying on a small anvil. A fair sample of the whole lot of gold dust under examination should then be placed in an evaporating dish, the suspected pieces being placed on top, and nitric acid poured over them. If any reaction takes place, such as effervescence, or evolution of red fumes, or if the acid becomes coloured, there is foreign matter present, and should this be the case, adulteration or counterfeit gold dust may be suspected.

“Place two watch glasses, one on a piece of white paper and the other on black, or some dark colour ; then, with a glass rod, convey a few drops of the acid from the dish to each. To the white add a drop or two of ammonia, until it smells strongly ammoniacal ; a blue colour indicates copper. To the other add hydrochloric acid in the same manner. If a white, curdy precipitate forms, which does not dissolve upon the addition of water, silver is being dissolved from the gold dust in the evaporating dish. If the dust is of very low grade these metals may dissolve in very small quantities. But such gold dust would be easily detected by its inferior colour and appearance.

“If no action is observed, even after heating the dish, there is no counterfeit present. Counterfeit gold dust is sometimes so heavily coated with pure gold (by the galvanic process) as to protect the base alloy from the action of nitric acid, hence the necessity of cutting all suspected pieces before submitting to the action of the acid. To remove the acid from the gold, wash with water thoroughly, and dry over the spirit lamp.”

CHAPTER III.

WORKING SHALLOW GRAVEL DEPOSITS AND THE BEDS OF RUNNING RIVERS.

THE preceding chapter has been devoted to an account of the geological localities of auriferous formations, coupled with some opinions as to their origin and cause : it has, in fact, dealt with the subject in its broad general outlines. The next consideration will be the specific characters of the several formations: and the means employed in extracting the gold from them.

Definition.—The term “shallow placers” is applied to auriferous deposits in alluvial ground overlying the “country-rock,” and reaching to depths varying from a few inches to many feet. Such are always of recent formation, as opposed to the so-called “deep leads,” to be considered in the next chapter. The working of these deposits is often known as “surfacing.”

Importance.—It was in these superficial accumulations that gold was first found by colonists near the foot-hills of the Sierra Nevada, as well as in the creeks and gullies of Australia ; and so rich were they, that even with the primitive appliances in vogue in the early days of the gold industry, 36,000,000*l.* are said to have been got out of the Californian diggings during the first 5 years' work. In fact, until quite recently, since such great progress has been made in quartz mining, at least two-thirds of the gold produced in English-speaking lands has been by washing this superficial *detritus*, and all the Siberian product to this day is obtained from the same source. The reason for this is not far to seek.

Formation.—In these shallow diggings, Nature has for ages been performing the work for which the quartz miner must invent all manner of machinery, and employ a vast amount of capital and skilled labour—the disintegration of the gold-bearing rock and the concentration of the metal. Consequently, the unskilled labourer, whose capital is represented by his own strength and a few of the simplest possible tools, is able to extract on a remunerative scale immense quantities of gold, which under its original condition, disseminated through quartz and other hard rocks, often in invisible proportions, would have needed vast amounts of capital and much machinery for its elimination, and would in many instances not have repaid the outlay.

Decline.—The exhaustion of the shallow placers of the older gold-fields is fast approaching, that class of mining being abandoned to the "more patient though less skilful Chinese." In New Zealand, surface washing is now almost entirely a Chinese occupation, the careful Celestials being satisfied with the leavings of Europeans, and content to wash the tailings whence the bulk of the gold has been extracted.

But, besides the enormous yield of gold from the shallow washings themselves, which gave men confidence and capital necessary for undertaking the less obviously beneficial exploration of deep leads and quartz veins, they have actually led the miner up to the very doors of these hidden store-houses of wealth. In New South Wales, as in Victoria and California, the deep leads have nearly all been discovered by prosecuting the surfacings on the hill-slopes, and indeed in many instances the outcrops of quartz veins have been accidentally discovered in placer-diggings by teamsters and other similarly unscientific gold-seekers.

Though placer-mining, for many reasons, does not now occupy the leading position it once held, it must still remain highly important, and will continue to take the foremost place in all new gold-fields, whether in old or new countries. For this reason, it is fully deserving of the attention here bestowed upon it.

Characters.—The origin, occurrence, and character of shallow placer-diggings vary considerably. Von Cotta has laid down half-a-dozen rules deduced from the method of formation of placer-diggings which have been singularly verified in practice :—

1. Placers *in loco* will be likely to carry metals in quantity and distribution like the original deposits on which they lie.
2. Alluvial placers of accumulation will be richest in those places where the current of the stream was interrupted by a diminution in its fall, by sudden change of direction, or by the entrance of a tributary ; also by reefs, bars, and eddies. The absolute richness, however, depends upon local circumstances, and the size and weight of the floated masses must be taken into consideration.
3. Of course the small depressions, creases, holes, and fissures of the bed-rock over which the current passed are frequently especially rich.
4. The lowest layers of each "period of deposition" (*Schwemmpériode*) are usually the richest.
5. Sometimes, however, several periods of deposition have succeeded each other ; and thus several rich strata may occur in the same ground.
6. Not only the courses of present streams, but also, and especially, the ancient channels, now forsaken, are the localities of placers.*

* This paragraph refers to deep leads, which are described in the next chapter.

A "period of deposition" may be understood as the time during which all the mass of alluvial material was subjected to the influence of water. By the cementing process which will be presently described, or by an interruption in the aqueous action, the alluvial layers may become so consolidated as to form an apparently new bed-rock, then termed a "false bottom"; or the accumulation in depth may even be such that the stream fails to have any effect upon the bottom stratum. Wherever shallow placers have been worked, experience has shown that there are frequently two or three "bottoms" and corresponding strata of maximum richness alternating with others comparatively barren. Many old diggings in all parts of the world, supposed to have been quite exhausted, will yet be worked and yield as great riches as before, when this fact is thoroughly appreciated and acted upon.

There cannot be a doubt that in mechanical and not in chemical processes have originated nearly all alluvial gold-deposits, that they have been primarily derived from the disintegration and wearing away of auriferous veins of quartz, &c., exposed to the influence of the weather and other agencies, at or near the then surface of the ground. As a rule, the *detritus* thus formed has been distributed by the power of running water; but that has not always been the case, for sometimes placer-diggings are found on the very outcrops of auriferous veins or reefs, constituting what Von Cotta calls "placers *in loco*." These last, not having undergone any appreciable amount of mechanical concentration and accumulation, are generally, if not universally, much less rich in metal than the placers where the concentrating agency of a stream has been at work.

Some placers even are attributed to glacial action. Thus Professor Kerr, the State Geologist of North Carolina, is of opinion that all the auriferous gravels of the State have been formed by what he calls "frost drift" or "earth glaciers," i. e. the effects of repeated frost and thaw in decomposing the rocks, and then, by alternate expansion and contraction, causing their detritus to rearrange its component parts. According to Kerr, there are probably 500 square miles of gold-drifts in North Carolina. They are found through a range of 400 miles east and west from the lower waters of the Roanoke, near Weldon, to the extreme western border, the county of Cherokee; and they belong to all the different subdivisions of the Archæan rocks of the State. The two most extensive deposits are found in the middle region on the Yadkin and Catawba rivers, among the low ranges and spurs of the mountains. The schistose and slaty rocks, highly inclined and much contorted and dislocated, are in many places penetrated by innumerable small veins and seams of gold-bearing quartz. In the disintegration and breaking-down

of these rocks, and the movements of the *débris* in the manner described, it is evident that the gold particles, with the heavier crystalline minerals, will be accumulated near the bottom of the drifts, on or near the surface of the bed-rock, or "slate," as the miners call it.

Touching the glacial theory, it will be interesting to quote a recent report of the State Mineralogist of California. He says:—

"When I made the discovery, by the use of the microscope, that all of the sands in the hydraulic mines were angular, and not rounded by the action of water, as I expected to find them, I came to the conclusion that the river beds had not been filled by force of water, as was generally supposed, but that we must look for a new theory based on the new discovery. I naturally looked to ice as the agent, and attributed the filling of the beds and the disintegration of rocks, to the action of glaciers moving over the land. This view, while it would account for many of the phenomena, did not account for others; the rounded boulders were a stumbling block which could not be overcome, and their formation, by long continued action of water, could not be made to harmonise with the angular condition of the sands. With the glacier theory, also, it must be assumed that climatic conditions in the past must have been very different from now, of which we have no proof. When I discovered that the boulders were probably formed long before they were deposited in the river beds and cañons of earlier times, a new light dawned upon my mind, and I began to think that the great lava flows might be the active agents in filling up the inequalities of the land, and that the boulders already formed had been picked up by the flow or pushed forward by the approaching lava mass. This was a striking feature in the new theory, and only required the lava cold to be perfect. Soon after, I visited the Spring Valley Hydraulic mine, at Cherokee; here I found the formation cut down for 430 ft., exposing a gigantic section, from the lava cap of basalt to the bed rock of syenite. Studying this section carefully, I noticed matter in the form of indurated mud, or semi-lava, in which were imbedded rounded boulders, of a similar nature but of a different colour, suggesting the idea that the former may have flowed, in a semi liquid state, and gathered up the soft boulders as it moved; this idea called to my mind what I had noticed with wonder in other localities.

"At Picket Post, in Pinal County, Arizona, there is an immense formation of this character, which covers the country for many miles. I had the pleasure of visiting this locality, in company with Professor Cook, State Geologist of New Jersey, and we speculated as to the origin of the deposit. We noticed that the so-called lava partook more of the nature of plastic earth, or volcanic mud, and that it had a brecciated

appearance, as if it had picked up boulders of a similar but older formation. We noticed, imbedded, rounded boulders, or pebbles of obsidian, in the mass, for which we could not account.

"On the borders of Mono Lake there is a lava flow which partakes of the compound character of true lava and of volcanic mud, which, in a like manner, is filled with rounded boulders of a different formation.

"Before visiting the Cherokee mines, I had seen the Indian Spring drift mine, and followed the bedrock in the tunnel 1730 ft., to a locality far below the lava, where, a short time before, the tusk of an elephant and tooth of a peccary had been found, now preserved in the State museum. I wondered that these gravel deposits were always of the same character and invariably capped with basalt, except where denuded by more recent geological forces, and sought in vain for a solution. I was exceedingly fortunate in having my attention called to phenomena, now in full action, which threw a new light on the subject.

"From Cherokee I was invited by Mr. Louis Glass, Secretary of the company, to ride over the surface of the Table Mountains with him, and to visit the hydraulic mines in Morris Ravine, owned by the Hon. W. C. Hendricks. As we rode over the level lava plain, it became clear to me that a flow of lava had swept over the face of the land, filling up all the inequalities, and forming a level plain, cutting off at the same time all the outlets of the ancient streams, which immediately set about to cut new channels, and, in course of time, forced their way through the lava, forming, in time, the valleys before mentioned.

"On reaching Morris Ravine, I witnessed a strange sight. Here was a moving mass of earth, miles in extent, governed in part by the same laws which apply to glaciers. When Mr. Hendricks commenced hydraulic mining in the ravine which has been singularly rich in gold, he met with success. As he progressed he noticed certain singular phenomena for which he could not account. In one portion of the claim the ground was found to have risen, while it had sunk at others. When piping to remove the auriferous earth, he noticed that he did not seem to progress, or to uncover the bedrock to the extent expected, when, at last, it occurred to him that the ground was slowly moving forward, in proportion as the earth was removed by the powerful hydraulic stream. When he fully realised this, he watched more closely, and found it to be a fact. Strange as it may seem, here we have many of the conditions which produce the glacier, but the yielding mass of matter is earth instead of ice. The gravelly deposit lies on a sloping bedrock, the inclination of which is not great enough to produce a land-slide, but sufficiently so to cause the flow, so to speak, in the direction of the least

resistance ; and this wonderful earthy glacier (if such a term is admissible) has crawled forward for years, and, although the motion is invisible, it still continues, and will continue until it reaches a point of equilibrium. While this is a wonderful and interesting geological phenomenon, it has proved a great misfortune to Mr. Hendricks, who can see no hope of profit in washing away the surface soil, containing but little gold, which is replaced as fast as he can remove it. Here is a striking instance of the fact that common earth and rocks yield to the force of gravitation and pressure, and move for considerable distances without the assistance of water, and without being fused, as in the case of lava. This will become an interesting locality to the geologist, and will be more carefully studied in the future.

"After carefully considering the above facts, I feel at liberty to form them into a new theory as to the old river beds and their auriferous gravel fillings, subject to new discoveries and to the criticism of geologists. The facts cannot be disputed, while the theory is likely to be. But I trust it will lead to a fuller investigation of this most interesting and important subject, and the discovery of new facts bearing upon it.

"My theory assumes that, at a period not very remote, geologically speaking, the ancient rivers flowed in similar channels to those since cut by the modern ones ; that the bedrocks contained veins of gold-bearing quartz, as similar bedrocks do now in the mountain regions, above the gravels. Fragments of rock had weathered into boulders, which lay, as they do now, throughout the mountain region. An eruption of volcanic mud flowed from an unknown crater, followed by, and pushed forward by, an eruption of igneous lava. As the volcanic mud flowed over the land, it gathered together the boulders lying on the surface, to which was added the rocks crushed and ground to a powder by the moving mass, by which all the irregularities of surface were filled up to a level plane, over which flowed the basaltic lava. By the erosive force of this slowly moving mass, the surface rocks were ground to a dust and the gold set free, which, with that already freed by former geological forces, was gathered together in a confused mass, as we now find it in the hydraulic and drift mines of the State. This deposit, instead of being the work of one eruption, may have been the combined labour of many, and ice may have played an important part. But if the grinding force had been wholly ice, it would be difficult to account for the universal presence of lava as a cap for all."

It must not be forgotten that the erosion of gold-bearing mineral veins, and the distribution of the drift produced, has taken place at two distinct geological epochs, both comparatively recent, but separated by many ages of time. The first series in point of age includes the so-called "deep leads" (described in the next chapter), and is the work of a river-

system quite different from that now existing. The second series, of which we are now particularly treating, is entirely due to the streams of to-day, and is being continued by them at this moment. The modern rivers in cutting through (almost always crossing, and frequently at right angles to) the channels of the long extinct water-courses, have redistributed their golden sands and gravels, and in this all the modern placers have immediately originated, except those few which are derived directly from mineral veins.

As may be gathered from what has already been said, placers are generally much richer in their richest parts than the veins from which they have been derived. Moreover, there appear to be good reasons for supposing that the gold-dust, under favourable conditions, will amalgamate and form nuggets such as are rarely or never found in veins. But, under certain circumstances, the veins may be more productive than the placers to which they have given rise; for example, if the auriferous rock be so hard as to resist disintegration more effectually than the country-rock, so that for a very small amount of the latter eroded there will be an immense mass of the former. But such instances are very rare. On the other hand, however, the amount of material to be removed for obtaining a given quantity of gold is generally much greater in placer than in vein deposits, especially in the case of deep leads and hydraulic workings.

Numerous instances might be given where shallow placers are entirely due to the degradation of quartz veins. In all such cases, the drift will prove barren above the point where the reef crosses it. In some placers in the Sierra Nevada, there is very little alluvium or drift, the gold having all come from ledges near by, which contain auriferous veins. The occurrence of gold-placers below the outcrops of silver-mines is illustrated by the Comstock ledge, which was discovered by following up the alluvial gold-workings to their source. This vein has ever since produced a highly auriferous silver-ore, but argentiferous ores much less rich in gold may in time produce valuable gold-placers, because the silver rapidly disappears by oxidation and solution, while the gold remains unattacked.

It will be appropriate here to call attention to recent evidence tending to prove that placers are sometimes formed by the infiltration of water containing very small quantities of gold in solution. This water having percolated through from the surface of the ground, the gold has been thrown down by organic matter, either dissolved in the water or scattered in a solid form through the materials constituting the bulk of the alluvial deposit. Already on p. 62 an account has been given of very extensive placers in Africa, which are declared by experts to have had their origin in this manner.

In character, placer-diggings manifest almost as great variety as vein deposits. In one case, on the Ballarat gold-field, Victoria, the "wash-dirt" or auriferous alluvium runs in a series of leads of varying width, starting from nearly the same point, and trending in different (sometimes opposite) directions towards the deep leads. In another placer, on the same field, the width of the "gutter" and "reef-wash" was about 100 ft., the depth of the "pay-dirt" about 5 ft. The "leadings" or barren drift overlying the pay-dirt was of black clay, the reef of green slate, and the bottom of sandstone. At a third, the wash-dirt, 4 to 6 ft. thick, was a loose dark-coloured gravel, intermixed with black conglomerate and sandstone boulders. In a fourth instance, the wash-dirt on one lead was a dark-blue or black gravel on a green slate bottom; but there was a second and shallower lead alongside, where the wash-dirt was much lighter in colour. This latter was supposed by some to have been the original water-course, and the former to have been caused by the stream being displaced from its proper channel. By others, it was regarded as a "reef-wash." The Wood's Point district, lying about 90 miles north-east of Melbourne, has some interesting peculiarities. The formation is schistose, the rocks belonging to the Upper Silurian series, and it is traversed by dykes that have cropped out on the surface and are generally richly metalliferous. The valleys are very narrow, and the beds of rivers and creeks are rockbound, and rarely exceeding 20 yd. in width. Deep leads are entirely wanting; the wash through which the alluvial miner works down to the beds of the creeks consists invariably of layers of water-worn stones, varying in size from that of boulders to that of pebbles, and is 2 to 12 ft. deep. The gold lies sometimes, but rarely, in a thin layer of sand or pipe-clay on the surface of the bed-rock; more generally, in the crevices of the rock itself, often more or less rotten, which is broken up to a depth of 12 to 20 in., and is occasionally found in what are termed "pot-holes," from the fact that they are of the form and size of a camp pot, say 15 to 18 in. in diameter and 6 to 10 in. deep. In the latter instance, the bed-rock is either a very hard blue schist or soft rotten granite.

In the first instance mentioned, the course of the stream is generally across the "strike" of the rocks, and the gold is found below a hard "bar," as shown in Fig. 5, or on one side of the creek, as in Fig. 6. In the second and third instances (Figs. 7, 8), the stream generally runs with the strike of the rocks, or at a slight angle; but the "dip" is nearly perpendicular in those instances where "pot-holes" have been known to occur. The reference letters indicate as follows. Fig. 5: *a*, surface of stream; *b*, mud and gravel forming bed of stream; *c*, bed-rock; *d*, auriferous gravel retained by the projection of the bed-rock. Fig. 6: *a*, bank of stream; *b*, auriferous gravel accumulated in deepest part of stream;

c, mud and other worthless matter lying on the pay-dirt. Fig. 8 : *a*, stream ; *b*, mud and gravel at bottom of stream ; *c*, bed-rock ; *d*, pot-holes in bed-rock where auriferous material has lodged.

FIG. 5.

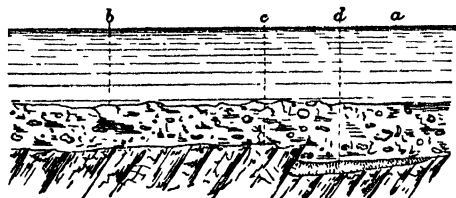


FIG. 6.

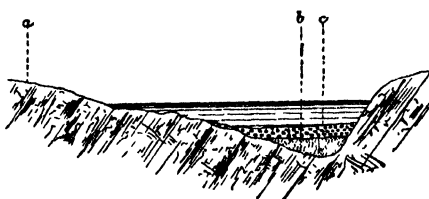


FIG. 7.

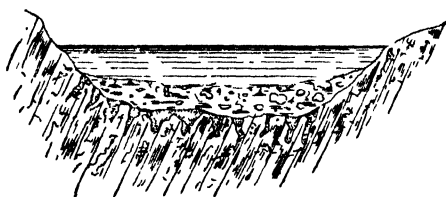
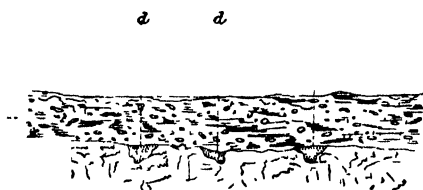


FIG. 8.



SECTIONS SHOWING SHALLOW PLACERS.

The alluvial gold on these bank workings or creek beds, as already stated, is found chiefly in the bed-rock, which is slate, striking north-west and south-east, and dipping nearly 90° . Some of these slates are soft and rotten to a depth of 3 ft. or more ; others are indurated and crystalline from the surface downwards. These differences occur within 50 yd. or less, and it is on the soft rock only that the precious metal is found. The gold is scaly or flattened. Nuggets are obtained in soft clay lying on the face of the bed-rock, at no great distance from it, say 3 or 4 ft. at most. Whether found *in situ* or drifted, they are believed to be of later date than the reef-gold, or gold found in the bed-rock. In the Goulburn, the bed-rock is commonly covered by a soft layer of heavy slate and granite boulders, in pipe-clay, some 2 to 6 ft. deep ; and above this is frequently found a bed of clay and pebbles, a few inches to 2 ft. thick ; and then a drift composed of large flat stones, much water-worn, cemented with clay and ironstone grit, sometimes 6 to 8 ft. thick. The stones are flat, weighing as much as 2 cwt., and elongated, their longer axis lying with the current, and their pointed ends up the stream. The last two drifts mentioned are sometimes absent, and in their stead is a heavy bed of red clay, with small quartz and slate breccia. The "colour" is got in all the drifts, but the best results are obtained from the bed-

rock, from which may perhaps be inferred that the chief and earliest deposits of gold in these ranges are contemporary with the first great denudation of the granitic dykes, which are much more numerous than at present appears. At the heads of the water-courses thereabouts, granite is never entirely absent, and the miners look for a greasy wash near the bottom as the most favourable symptom. Small deposits of wash are found high up on the spurs even to 1200 ft. The gold lies in these cases in a soft clay, and is "shotty" and without quartz. Nearer the bed of the river, however, where some of the bank claims have proved very rich, the gold is again found chiefly in the slate; and the drift above the bed-rock is composed of slate boulders and clay, the granite boulders being less frequent and not so large as in the bed of the river.

It is a curious fact that deep pools under waterfalls in auriferous streams seldom contain an appreciable quantity of gold. Many such have been worked out in expectation of a rich harvest, but with disappointing results. All large auriferous rivers show analogous cases; in them the gold is always found on the bars or points, and not in the deep pools or bends.

Sections of Strata.—Brough Smyth, from whom much of the information in this and the succeeding chapters is derived, justly observes with regard to sections of the strata found in placer mines that, though they may be and are generally uninteresting to the geologist, they are nevertheless highly valuable to the miner, and that if there had been many such sections procurable at the time when the placer-diggings of Victoria were first opened, much useless expenditure of time, capital, and labour would have been avoided. No apology, therefore, is needed for introducing a few sections here, principally from the Victorian fields:—

	ft. in
1. Original surface—dark loam	6 in. to 1 0
Sand, with very small fragments of slate and quartz, very sharp and angular	1 ft. to 2 0
Compact sandy gravel, changing gradually to quartz gravel, particles increasing in size towards the base of the stratum	5 0
Sand and gravel, principally sand, without quartz pebbles	1 0
Auriferous drift—gravel, and with very fine quartz, and a small proportion of sandstone intermixed with large quartz boulders, 6 to 12 in in diameter	1 ft. to 5 0
Decomposed bed-rock, very compact white clay	9 in. to 1 0
	15 0
2. Surface soil, say	0 9
Sand mixed with fragments of quartz and sandstone (not water-worn)	1 ft. to 2 0
Hard dark clay	0 3
Carried forward	3 0
	L

	ft.	in.
Brought forward	3	0
Red and yellow indurated clay in alternate layers of about 2 in. in thickness	1	0
Gravel intermixed with fine quartz	6 in. to	1 0
Layers of red and yellow indurated clay, sand and soft sandstone in alternate layers, 3 to 4 in. thick	1	0
Fine gravel with fragments of angular quartz	0	6
Auriferous drift—coarse gravel with fine quartz, and a small quantity of sandstone (not water-worn), and large quartz boulders ..	3 ft. to	4 0
Decomposed bed-rock, pipe-clay	6 in. to	1 0
	<u>11</u>	<u>6</u>
3. Surface soil, say	1	0
Hard sandy soil slightly mixed with gravel and a small proportion of very fine angular quartz	8	0
Auriferous drift—loose gravel intermixed with fine quartz and sandstone in small proportion, and large quartz boulders	8 in. to	1 0
Decomposed bed-rock, pipe-clay	6 in. to	0 8
	<u>10</u>	<u>8</u>
4. Surface soil, say	1	0
Sand mixed with angular fragments of quartz and sandstone ..	1 ft. to	2 0
Hard dark clay of reddish-brown colour	4 ft. to	5 0
Loose gravel intermixed with fine quartz	8 in. to	1 0
Loose sandy gravel, more compact in the lower part	5 ft. to	6 0
Sand, gravel, and quartz	2	0
Auriferous drift—loose gravel with sand and small angular fragments of quartz and schist	3 ft. to	7 0
Decomposed bed-rock—pipe-clay	0 in. to	1 0
	<u>25</u>	<u>0</u>

Of this last drift, it is remarked that the ground gradually deepens as it is followed northwards, until it becomes a "deep lead." In some parts, it would seem that the older drift has been much denuded, and only the lower layers left, upon which newer auriferous drifts have been superimposed. The miners not infrequently penetrate a "false bottom," which is covered as usual with rich wash-dirt and, after sinking through sand, gravel, and clay, reach another layer of wash-dirt immediately overlying the true bed-rock. It is only by a careful survey, and noting the sections, that it is possible to say how many auriferous layers may exist in an alluvial digging.

	ft.	in.
5. Surface soil	2	0
White indurated clay	20	0
Light-brown sandy gravel	1 ft. to	5 0
Conglomerate red and brown, and water-worn quartz	1 in. to	0 4
Wash-dirt with rough reef-gold mostly attached to fragments of quartz	6 in. to	2 0
	<u>29</u>	<u>4</u>

	ft.	in.
6. Loam	4	0
Red clay and gravel	13	0
Reddish clay	18	0
White clay and sand	12	0
Red gravel and sand	8	0
White clay and gravel	7	0
	62	0

The wash-dirt, composed of white clay and gravel and small pebbles of quartz, is about 1 ft. in thickness.

7. Yellow clay on the surface, with layers of red and white gravel intermixed with a heavy wash of boulders and gravel on the bottom—composed of slate and soft pipe-clay (decomposed mudstone). The wash-dirt taken out at the first rush varied from 1 to 3 ft. in thickness, but since then many parts of the strata reaching from the surface to a depth of 20 ft. have paid for washing.

8. The depth of the shaft was 85 ft., through stiff clay, gravel, and cement. The wash-dirt was white gravel, intermixed with heavy boulders on a soft pipe-clay bottom; its thickness being 2 to 5 ft.

9. Top soil	0	6
Red clay	7	6
Quartz gravel and red clay	7	0
Greyish-coloured sand	6	0
Gravel and quartz boulders, ironstone and pieces of slaty rock cemented with clay	8	0
Fine red gravel and ferruginous clay—auriferous	2	6
Soft yellow-coloured clay-slate	..	..

10. Top soil	1	6
Red clay	6	0
Red gravel and cement	8	0
Sandy clay and gravel—wash-dirt	1	0
Argillaceous schist covered with pipe-clay	..	..
	16	6

The greatest depths mentioned by no means indicate the maximum limits of the thickness of placer diggings, for shafts have been sunk through similar strata to the extent of 300 ft. without reaching the true "bottom" or "bed-rock." In the Daylesford division (Victoria) the depth varies from 1 ft. to 150 ft., but little being over 50 ft., and in one quarter (3 months) 33,560 tons of dirt were washed for an average yield of only 18½ gr. per ton; in the Majorca division, during the same period, 51,200 tons, got from 170–175 ft., gave 3 dwt. 3 gr. per ton; and in the Maryborough division, 14,500 tons, raised 270–275 ft., afforded 2 dwt. 10 gr. per ton.

Conditions, favourable and unfavourable.—The first great desideratum in placer working is water, for mechanically removing the valueless

constituents of the "pay-dirt" or "wash-dirt," i.e. the auriferous stratum, and leaving the gold behind. Without the possibility of obtaining an abundant supply of water, diggings of high promise may be quite unavailable.* The merits of an auriferous patch may therefore be summed up in a great measure by the ease and cheapness with which a sufficiency of the element may be obtained. But a second and equally important point is the disposal of the "tailings" or waste material from which the gold has been extracted, or which has had to be removed in order to reach the gold-bearing layer. The neglect of this second point has often entailed fruitless expenditure. Of course, both conditions apply with redoubled significance to hydraulic mining; but it may be well to cite a few instances proving their weightiness even in shallow placers.

In the Government Reports on the gold-fields of New Zealand, for example, are to be found such remarks as the following:—

"Sluicing operations are brought to a standstill from a dread of damages consequent upon the deposit of tailings" on other people's property.

Again, "The ground thus occupied has long been known to be auriferous, but for want of fall it could not be worked. None of the claims can be wrought until the opening of a channel affords the long-desired means of sending away the waste water and tailings from the workings."

The falling-off in production at another diggings is attributed "principally to want of water"; and again, "the claims can only be worked by running the tailings through private property, and in course of time the lowest-lying portion of the ground must be covered up with the waste of sluicing if the workings continue."

On the other hand, floods are equally injurious, and, where prevalent, must be provided for by what is known as a "storm-channel," otherwise the whole "plant" on a diggings may be carried away in a few hours.

In Australia, too, flatness of the ground is a great obstacle in some cases, though not necessarily insuperable.

Similarly, in the Western States of America, the depression in placer-mining in 1870 was caused in a great measure, according to Raymond, by "the filling and choking up with tailings, after a few seasons of washing, of the ravines, gulches, and rivers which served as outlets from the ground, thus preventing the maintenance of 'flumes' of sufficient length and grade to disintegrate the hard cement found underlying the top dirt, and rendering imperative the construction of long tunnels to the nearest deep stream, requiring, in some cases, years of

* Some plans of "dry washing" are described on p. 197, but they do not seem to have achieved great success.

labour and the disbursement of large sums without immediate returns." So in another instance—"The future of the region under consideration will depend to a great degree upon finding an outlet for its vast quantity of hydraulic dirt. This can be obtained in some places by 'bed-rock tunnels'; but at other points these are impracticable, and effectual accommodation can only be had by emptying the streams and gulches of their accumulations of tailings." A prominent example of the extent of these accumulations is the Bear river, in California. "This stream has been filled to a depth of nearly 80 ft. in the centre, and its former banks so far covered that tall pine-trees, formerly far above the stream, have been gradually engulfed season by season, until now only the top branches appear above the current." The quantity of tailings here, estimating "an average width of 300 ft., a depth of 75 ft., and a length of 10 miles, would be 44 million cub. yd. It is believed that these tailings contain enough gold and quicksilver to pay a handsome profit for their removal, if an outlet could be found."

The presence of bed-rocks cropping out on the surface is a valuable guide to explorers, enabling them to follow the trend of such alluvial patches as they may discover. As a rule, the gullies or little rivulets having their sources near the outcrops of auriferous quartz veins may be expected to be rich. Victorian miners believe that hills with quartz gravel and quartz boulders on the surface are most likely to prove auriferous.

Influence of bed-rock.—It will have been noticed already that the character of the bed-rock exercises an important influence upon the nature of the deposit. Slate forms one of the richest bed-rocks, as the gold accumulates behind the natural "riffles" or checks produced by its edges. In some of the chief diggings in Victoria, it has been noticed that hard sandstone bottoms were the richest; and in many cases, the gold has been found to penetrate as deep as 3 or 4 ft. in narrow crevices. The slates, and decomposed mudstone and clay bottoms were less rich in comparison. In one case, the gold in the deep ground was quite black, while that from the shallow parts was clean and bright.

Object in placer mining.—The object which has hitherto guided the operations of the placer-miner has almost always been to take the cream, if one may use such an expression, and to leave the skimmed milk; to hurry through the ground, taking out the bulk of the gold with the greatest possible economy of time and labour, and to let the rest go. This is one great reason why the patient Chinese can make a living out of ground that has been abandoned by Europeans. The evils of such a practice are self-evident: very much of what is left is so disseminated, and the ground is rendered so unworkable, that it is doubtful whether it can ever be recovered. Men, are now, however, beginning to realise the

importance of taking a little more care to save as much as possible of the precious metal, and the various improvements in machinery, &c., which have been attempted, especially for catching the fine gold, will presently be described in detail.

Principle of gold-washing.—The whole theory and practice of separating alluvial gold from the earthy matters and minerals with which it is found is dependent upon the high specific gravity of the precious metal; and all apparatus used in the operation is constructed and arranged on the principle that, while water has the power of removing the base material, the gold is almost entirely left behind.

Pans and panning.—It is by no means uninteresting to observe the widespread knowledge of the art of gold-washing among the native races of far distant countries, and by peoples not generally remarkable for scientific attainments.

The style of panning described by Mungo Park in no way differs from that now practised on the Gold Coast and Kaffir lands. It is illustrated in Fig. 9, and is conducted as follows. A woman puts about

FIG. 9.



GOLD-PANNING IN WEST AFRICA.

$\frac{1}{2}$ to 2 lb. of gravel and sand into the calabash shown, covers it 1 in. deep with water, and crumbles it to pieces and thoroughly mixes it by pulling her hands towards her. She then throws out all large pebbles, taking care not to reject nuggets at the same time; which done, she gives the sand and water a rotatory motion, so as to make a portion fly over the brim of the calabash. While doing this with her right hand, with her left she throws out of the centre of the vortex a portion of sand and water at every revolution. She then adds a little fresh water, and as the quantity of sand has by this time much diminished, holds the calabash obliquely, and makes the sand move slowly round on the line A B, while she constantly agitates it with a quick motion in the direction C D. The whole operation is completed in about 2 minutes.

The "pay-gravel" in Mexico lies deep below the surface (20 to 60 and even 100 ft.), and is generally very rich. Owing to the almost total absence of water, mining and washing have been but imperfectly conducted, and a large amount of gravel remains untouched. The Mexicans

sink circular shafts, like wells, through the soil and alluvions to the gravel ; tunnel upon the bed-rock ; take the good gravel to the surface in sacks ; cart it 2 miles to water ; and then pan out the gold in wooden bowls or *bateas*. In the winter, water is sometimes obtained by melting snow with heated stones.

The instrument used by the native alluvial gold-washer of the United States of Colombia, called *almocáfne*, is peculiar in its form and construction. The blade of this instrument has a semi-elliptical shape, is about 9 in. long and 2 in. broad at its base, with a gradual taper to the point, which is blunt. The shank, which is in a direct line with the long axis of the instrument, is cylindrical, and receives a handle about 8 in. in length. With this implement, which is made of tough, well tempered steel, the gold-seeker scrapes and digs among the boulders in the streams, and in the cracks and crevices of the strata everywhere. He has another implement for testing the quality of the earth or sand to be washed. It is a boat-shaped piece of horn, named *cacho de bateador*. In this he collects a small quantity of the material to be tested, and by repeated levigations with water, which is allowed to flow off afterwards slowly, the lighter and soluble portions escape, leaving behind the metal with its metalliferous associates.

When a locality is found sufficiently rich for washing operations, the loose earth, pebbles, and sand are scraped together, and if water is present, the larger pebbles and stones are carefully washed and thrown aside. The material thus collected is put into a wooden bowl (called by the natives *batea*), taken to a stream or pool of water, and there washed.

The *batea* of this region is conical in shape, and has 2 short projections from the brim, about 2 in. long, and opposite to each other, which serve as handles. The cavity of the bowl terminates below, in a point, but the outside is slightly truncated. The *batea* is made from the mountain cedar, and is light and strong. The method of using it is much the same all the world over.

At Jhunan, on the Mahanadi, Central Provinces of India, there is a small colony of gold-washers, who travel to the different localities when the water falls after the rains. Fig. 10, after Ouseley, illustrates the gold-washer's art as prosecuted at Heera-Khoond [*Híra-Khudá*], an island in the Mahanadi, between Sambalpur and Sonpur.

The class who more particularly follow the trade of gold-washing in India belong to a tribe of Gonds known as Jhoras, Dhoras, Dokras, Toras, or Jharas, according to locality. The name Jhora is said to be derived from *jhori*, a small stream, but Dhora may be derived from *dhona*, to wash. In the case of these people, both sexes engage in the pursuit ; but the Ghasis, a local tribe of unknown origin, also occasionally

wash for gold. Among them the men only, while among certain Kol or Munda tribes to the west of Singhbhum the women wash for gold, and their male relatives regard the work as an unworthy occupation for their sex. The methods employed by these different tribes appear to be identical in all essentials. Each occupies a distinct tract, and poaching on one another's favourite streams is not indulged in to any great extent.



GOLD-WASHING AT HEERA-KHOOND

The wooden dish used for washing measures on an average about 28 in. by 18 in. for the men, smaller ones being used by the women and children among the Jhoras. The dish is hollowed somewhat eccentrically to a maximum depth of $2\frac{1}{2}$ in. A scraper formed of a flattened iron hook, set in a handle, is used to collect the auriferous sand and gravel which accumulates in the angles formed by the rocks in the beds of the streams. The dish when filled is placed in shallow water, and the operator working with his hands soon separates and throws aside all the coarser gravel and stones, while the agitation of the water serves to carry away all the mud and lighter portions. The dish is then balanced on the palm of the left hand, and oscillated to and fro with the right; this serves to throw off the greater portion of the remaining gravel, and the process is completed by a circular motion, which is communicated to the water in the hollow of the dish, by which even the smallest particles of foreign matter are separated, and the final result is a residue of black iron-sand in which the specks of gold are readily apparent; but as

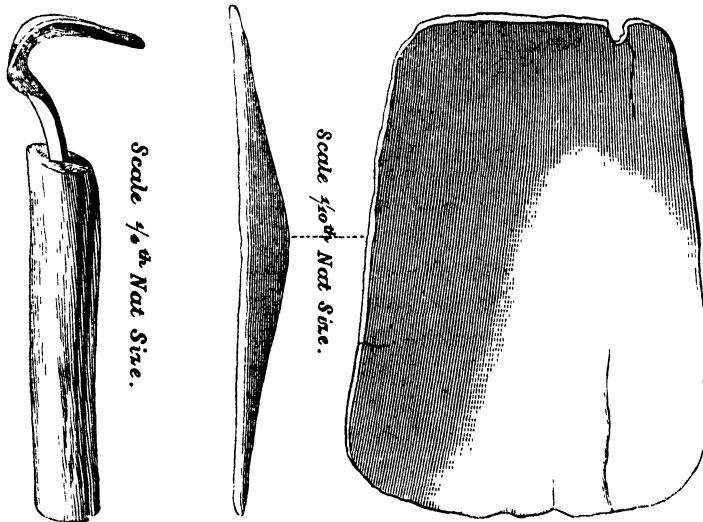
mercury is not employed in this part of the country, all the very small and invisible gold is lost. Fig. 11 will enable the reader to realise the

FIG. 11.



GOLD-WASHERS OF SINGHBHUM.

FIG. 12.



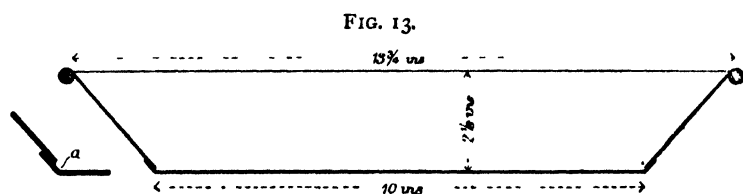
DISH AND SCRAPER USED IN SINGHBHUM.

appearance presented by these gold-washers when pursuing their trade, and Fig. 12 illustrates their apparatus.

The *batea* used in the Brazilian gold-washings and mines is a shallow circular plate, usually turned out of a single piece of wood, about 20 in. in diameter and $2\frac{1}{2}$ in. deep in the centre, the slope being gradual and regular to the outer edge. It requires peculiar manipulation and some degree of skill.

Another simple contrivance similarly applied is the horn spoon. It is made by cutting a piece obliquely out of a large ox-horn, so as to measure about 8 or 10 in. long and 3 in. wide, and is then scraped down to a suitable thickness. A horn that is black at one end makes the best spoon, as the gold is so much more readily seen against a black surface. It possesses many good qualities, and is a favourite among prospectors; the desired qualities are chiefly lightness, durability, and its not becoming enfilmed with air or grease, so as to prevent the perfect contact of the water on its surface.

The modern gold-digger's "pan" resembles a frying-pan without a handle (Fig. 13); indeed, in the early days of gold-washing, that humble household article was very frequently appropriated to the purpose. It is generally made circular in form, 10 to 14 in. in diameter at the bottom, but 3 or 4 in. wider at the top, as the sides, which are about 3 in. deep, are made to slope outwards to that extent. It may be made either of stout tin-plate or thin sheet-iron. The latter material is preferable, as well on account of its greater strength, as because it is not attacked by



GOLD-WASHING PAN.

mercury, which it is sometimes convenient to use. The best pans now employed in California are stamped out of one piece of Russian iron of the finest quality, and strengthened by a stout wire in the rim. At *a* is shown the junction between the bottom and sides of the pan, by which it will be seen that when the pan is not stamped out of a single sheet a small interval is purposely left between the floor of the pan and the lower edge of the side; this is in order to more readily retain the fine particles of gold when washing away the waste matters.

Simple as the process of panning appears to be, dexterity is only acquired by considerable experience. In outline it is as follows:—A quantity of the dirt to be washed is placed in the pan, sufficient to occupy about two-thirds of its capacity, and the pan with its contents is then

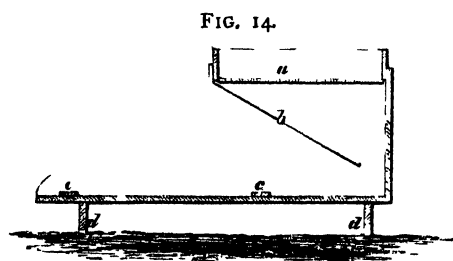
immersed in water, either in a hole or in a rivulet, of such a depth that the miner can conveniently reach the pan with his hands while it rests on the bottom. The object of this is to give the operator free use of both his hands for stirring up the mass, so that every particle may become thoroughly sodden and disintegrated. Of course the pan may be held in one hand, and its contents stirred by the other, but the disadvantages of such a method are obvious. When the dirt has become thoroughly soaked and permeated by the water, the pan is taken in both hands, one on either side, and a little inside of its greatest diameter, and without allowing it to emerge from the water, it is suspended in the hands, not quite level, but tipping somewhat away from the person. In this position, it is shaken so as to allow the water to disengage all the light earthy particles, and carry them away. When this has been concluded, there will remain in the pan varying proportions of gold-dust, heavy sand, lumps of clay, and gravel-stones. These last accumulate on the surface, and are picked off by hand and thrown aside; the lumps of clay must be crumbled and reduced by rubbing, so as to be carried off by the water during the next immersion of the pan. A neat turn of the wrist is required to allow the muddy matters to escape in dribblets over the depressed edge of the pan, without exercising so much force as to send the lighter portions of the gold after them. At last, nothing remains in the pan but the gold-dust, with usually some heavy black sand and a little earthy matter. By the final careful washing with plenty of clean water, the earthy matters can be completely removed; but the heavy iron-sand cannot be got rid of by any method based upon its specific gravity relatively to that of gold.

Removing iron-sand.—To effect this, one of two eminently simple plans must be adopted. If the iron-sand be magnetic, as is usually the case, it may be eliminated to the last grain by stirring the mass carefully with a powerful magnet, care being taken that no particles of gold become mechanically suspended among the black sand. Where this is ineffectual, recourse must be had to "blowing." For this purpose, the mass of gold-dust and iron-sand is allowed to become perfectly dry, and small quantities at a time are placed in an instrument called a "blower,"—a sort of shallow scoop, made of tin and open at one end. Holding the "blower" with its mouth pointing away from him, and gently shaking it so as constantly to change the position of the particles, the operator blows gently along the surface of the contents, regulating the force and direction of his breath so as to remove the sand without disturbing the gold.

Despite the many improvements introduced from time to time in gold-washing appliances, the pan yet remains an essential part of the gold miner's outfit. Many million pounds sterling were in the early days

extracted by its aid alone, and it is still used for finally panning out the rich mass of gold or amalgam collected in the various apparatus now employed. On new gold-fields, it continues to be the principal implement in use, as men are not able to carry machines to a "rush," and have no time to make them when they arrive at the scene of operations. As a receptacle for gold, amalgam, or very rich dirt, it is always handy.

Cradle or rocker.—The methods and implements chosen in working alluvial gold-diggings are almost entirely dependent upon the supply of water at command. The pan requires the least water of all, and properly stands first in the list of apparatus; but if sufficient water be available, it may be discarded in favour of the cradle. This apparatus is so called partly from its outward resemblance to an ordinary nursery cradle, and partly because it is provided with similar rockers, and is caused to oscillate in a like manner. Fig. 14 shows a longitudinal section of a



GOLD-WASHING CRADLE.

cradle, which is usually about 40 in. long and 20 in. wide, the back end 15 in. to 2 ft. in height, and the sides sloping down from that to about a couple of inches at the mouth. The movable riddle or hopper *a* fits exactly into the top of the cradle, so as to sit steady when the latter is in motion. It is about 20 in. square and 6 in. deep, with a

bottom of sheet iron closely perforated with $\frac{1}{2}$ -in. holes—at least that is the general size, though not necessarily adhered to when the pay-dirt is very fine. Below the grating, hangs a curtain or apron *b*, of canvas, blanketing, or other suitable material. This is made by stretching a piece of cloth, or whatever the material may be, on a framework, which is introduced from the mouth of the cradle, and rests on fillets on the sides, sloping from the mouth of the cradle towards the angle formed by the junction of the bottom and the back. Crossing the bottom of the cradle, are nailed a couple of "riffle-bars" *c*, one near the middle, the other towards the outside edge, and each about $\frac{3}{4}$ in. high. The apparatus stands on rockers *d*.

The mode of working with this apparatus is as follows. Some auriferous earth is thrown into the riddle, and the operator then proceeds to rock the machine with one hand while he pours water over the dirt with the other. Under these continued influences, the dirt disintegrates rapidly, penetrates the bottom of the riddle, and, falling on the apron, is conveyed to the inner end of the cradle floor, whence it flows back over the riffle-bars and out of the mouth, the cradle being placed on an

inclined plane when at work, so that the difference in level at the hopper end and the mouth end shall be commonly about $2\frac{1}{2}$ in.; but this is subject to modifications to suit special kinds of dirt and the fineness of the gold. Almost all pay-dirt contains more or less stones of various sizes, which will be retained in the riddle, unless small enough to pass through the grating. Those which are so large as to interfere with the working of the cradle are at once picked out and thrown aside, without checking the operation; but the smaller ones are allowed to accumulate, both because their removal in bulk wastes less time, and their presence in the riddle assists the process of disintegration of the earthy matters found in the gravel. When the hopper has become quite full of stones, all washed clean, they are tipped out and carefully looked over for any nuggets of gold which may be among them. The finer particles of gold will collect behind the riffle-bars on the bed of the cradle, while a certain amount of the very fine gold will be caught by the hairs of the blanket-apron. From these receptacles, the gold, heavy iron-sand, &c., have to be gathered periodically, the intervals depending upon the nature and richness of the auriferous earth under treatment. This proceeding is termed "cleaning up," and generally needs to be done two or three times a day. The hopper is removed, and the apron is then withdrawn and carefully washed in a bucket or other vessel containing clean water which dislodges the gold that may be entangled among the hairs, so that it can be recovered from the bottom of the vessel. Next, the gold and other matters which have collected behind the riffle-bars are scraped up with an iron spoon, and subsequently panned out.

As the weight of water required for cradling is at least three times as great as that of the material to be treated, it is better to carry the latter to the former than *vice versâ*. Of course to be able to convey the water by its own gravitation to the spot where the washing is to be done is of immense advantage. Care must be observed to have sufficient fall and outlet for the "tailings." The water may be conducted into a little pit to serve as a reservoir near at hand for the cradler to ladle it out, or it may be more conveniently laid on by pipes or wooden gutters so as to flow into the top of the riddle.

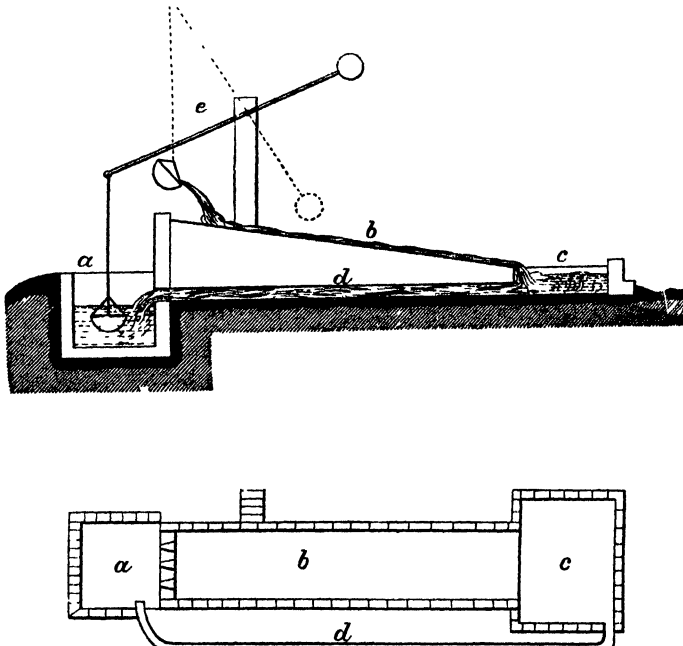
It is possible for one man to work with a cradle; but it is disproportionately inconvenient, as he has to be stopping repeatedly to introduce fresh supplies of dirt and to empty the riddle, and during these delays the sand will pack hard and fast behind the riffle-bars, and have to be removed each time before operations can be recommenced. Nevertheless one man can wash 1 to 3 cub. yd. daily, according to the more or less clayey nature of the dirt under treatment. A division of the labour between two men is much more economical, as one can be constantly rocking and supplying or regulating the supply of water,

while the second keeps the riddle fed with fresh dirt ; thus two men can cradle nearly three times as much as one.

Cradling is neither expeditious nor economical ; it loses fine gold, the sand has a great tendency to pack behind the riffle-bars, and its working capacity is not more than one-fifth of that of the tom and less than a tenth of that of an ordinary sluice ; but it is very cheap, requires little water, and is eminently portable. For these reasons, it is especially adapted for washing in gullies where the gold is coarse, and water scarce or uncertain, as is commonly the case in Australia, and there the cradle is still a valuable implement. It is a great favourite with the Chinese in California also. The use of mercury in the cradle is not advisable.

Where water is scarce, it may be necessary to use it repeatedly. In a case of this kind in Egypt, the Arab miners have adopted an ingenious

FIG. 15.



PLAN FOR RE-USING WATER.

plan, which is shown in Fig. 15, and may be adapted to almost any conditions. At *a* is a sump or water pit ; *b* is an inclined plane on which the mineral is washed, and whence the water escapes into a tank *c* ; *d* is a conduit for taking the water back to *a* ; *e* is a *chadou* or lever pump for raising the water. A certain amount of filtration could easily be managed during the passage from *c* to *a*.

Burke rocker.—The Burke rocker, of South Carolina, is longer than the ordinary rocker, and has a perforated plate of sheet iron, with holes $\frac{1}{2}$ in. in diameter, in the upper part, through which the fine materials fall into a riffle-box below, formed of shallow compartments, 5 in. apart, wherein mercury is placed. The machine, set on an incline, is rocked by means of a lever attached to the side, whilst a second person throws on the dirt, and removes the coarser gravel. It is said by Tuomey to do twice the work of the ordinary rocker.

Toms.—As with the modern pan so with the various forms of “tom” —their prototypes are to be found in use among the aboriginal inhabitants of many countries. The essential idea is embodied in an inclined plane surface provided with impediments behind which the gold will rest, while the lighter material is carried away by the force of the stream of water accompanying the pay dirt. Sometimes the inclined plane is covered with woollen cloth, to which the gold adheres: wanting the cloth, the Hungarian Gipsies now and then use, for the same purpose, the more ancient substitute of a fleece. The manner of collecting gold-dust in sheep's fleeces, upon inclined planes, is represented in the curious old work of Agricola. In the rivers of Colchis, the custom is still retained of placing sheep's skins in the beds of the Phasis and other auriferous streams, to collect particles of gold.

The Hungarian and Wallachian Gipsies exhibit great skill as gold-washers. Their implements consist of a plank of lime tree 2 or 3 ft. wide, 4 to 6 ft. long, and $1\frac{1}{2}$ in. thick, with 10 or 12 grooves cut transversely, and edged on both sides with a wooden rim. Woollen cloths are sometimes spread upon this board, which being held as an inclined plane, at an angle of about 45° , the auriferous sands are poured, mixed with water, upon it; the weightier sediment falls into the grooves, or it is retained by the cloth, which is afterwards washed in a water-cask; and then, by a common severing-trough, the sand is separated from the gold. But they are often skilful enough to collect auriferous pebbles, stamping them, and washing the powder. Fig. 16 shows the Gipsies practising their art. Generally they sell the gold thus found, in the form of dust; but some of them, who have been accustomed to work as blacksmiths, have ingenuity enough to smelt the gold into small ingots, using for that purpose, little low furnaces, and blowing the fire by portable bellows, made of buckskin. The construction of these bellows is as simple as it is ancient: they are made by fixing an iron air-pipe into the skin of the neck of the animal, and by fastening 2 wooden handles to that part of it which covered the feet.

The operation of gold-washing as conducted in the Ningthee river, Assam, is thus described by Captain Grant. The sand and gravel are first placed in a sieve, and the finer parts fall on to a hollow plank, 4 ft.

long and $2\frac{1}{2}$ ft wide at the upper end, and $1\frac{1}{2}$ ft. at the lower, which is open, the top and edges being protected by a rim or margin $\frac{1}{2}$ in high. The lower half is cut into grooves $\frac{1}{2}$ in deep and wide. The fine sand caught in these grooves is washed in a wooden dish, resembling a shield

FIG 16



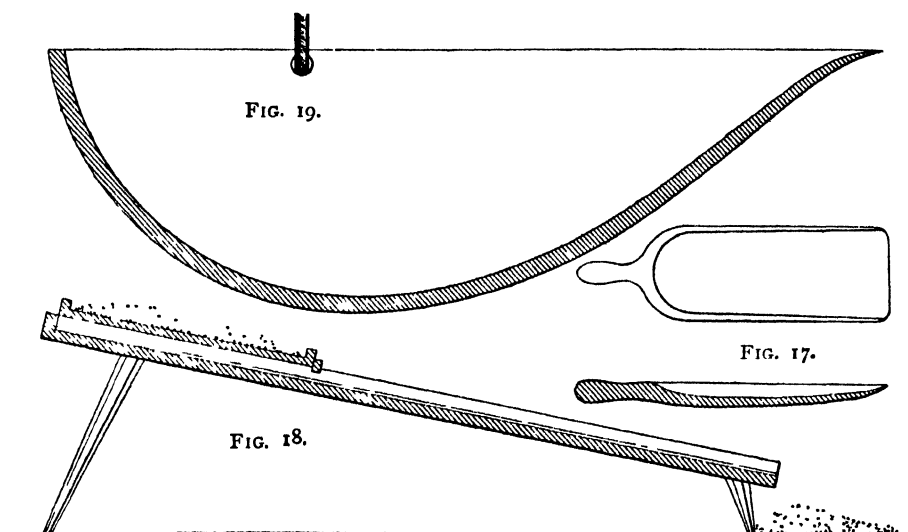
HUNGARIAN GITSIES WASHING FOR GOLD

in shape, which has a polished black internal surface, and a receptacle in the centre, placed floating on the water, it is revolved till all the sediment is removed, and the iron-sand and gold are left. By means of these implements, about 1 gr of gold is got in $\frac{1}{4}$ hour

The gold-washers or Julgars in the Dharwar district, India, use a sort of box made of light planks, which is 3 to 4 ft long, 20 in wide, and 9 in deep, for the first and rough part of the washings. The sand is then removed to a dish or tray in which it is panned off. The iron-sand with gold is further washed in a half coconut-shell, and the final residue is treated with mercury. Scholt speaks in high terms of the care taken by these men, which he says surpasses that of the Chinese in Australia, but Foote considers the process wasteful, particularly in consequence of the shortness of the washing-box. The earnings of these men during the season vary from Rs 5 to 50 (10s. to 5/)

The Rhine gold-washers use an instrument made of wood, charred black on the inside to facilitate discovery of the gold-grains, and of the shape indicated in Fig 17, measuring about $16\frac{1}{2}$ in in total length, $4\frac{1}{2}$ in.

in width at the mouth, and $\frac{1}{2}$ in. deep in the dish. The apparatus employed in washing the gravel has not varied from that described by Heberer in 1582, and illustrated in Fig. 18. It consists of an inclined table about 2 yd. long and 1 yd. wide, covered with a cloth made of long-stapled wool, set at an angle of 10° to 12° . At the head of the table is a sieve or hurdle made of osiers, the orifices in which are not more than $\frac{3}{4}$ in. in diameter; when this is filled with gravel, the operator washes it with water from a hand-bucket till all the smaller material has passed through the interstices, leaving the stones exceeding $\frac{3}{4}$ in. in



RHINE GOLD-WASHING APPARATUS.

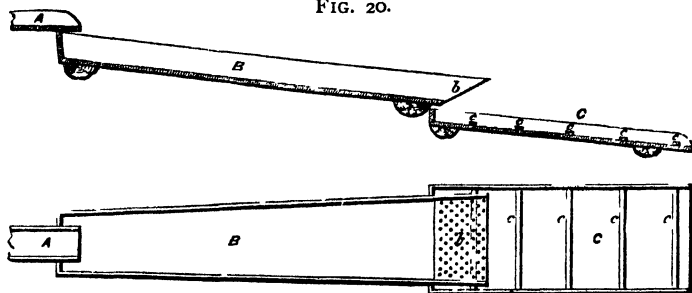
diameter, which are then thrown out. The fine sand and gold-grains remain for the most part caught in the wool, while the stones either roll away at once or are brushed out. When this has been done several times, the blanket is removed from the table and washed in a tub of clean water, by which the sand and gold-grains are dislodged from the fabric. This mixture of sand and gold is generally carried home to be separated, which is effected in the wooden boat-like vessel, termed a *Schiff* in Seltz and a *Sass* in Baden, shown in Fig. 19, and measuring about 4 ft. 6 in. long and 1 ft. 6 in. deep.

The cloth used on the table is that known locally as *drap de Souabe* or *Schwabentuch*, and is the same as the Tyrolese and the German waggoners used for cloaks. It endures about 1 year, if turned when the first side is worn out. The grains of gold are driven with some force by the water into the meshes of the cloth, and sometimes even completely

through, in which case they are arrested by a second cloth of cotton or linen, on which the blanket rests. This duplicate cloth is only necessary below the hurdle: the greater part of the gold stops at the lower edge of the grating. A man working 12 hours a day can treat about 4 cub. yd. in this manner. About 10 per cent. of the gold is lost in ordinary gravel at this stage; it might be diminished by reducing the angle of the table, but that would also lower the daily outturn. The gold collected subsequently in the *Schiff* is amalgamated with about $\frac{1}{4}$ its weight of mercury.

The old-fashioned "long-tom," now probably extinct, was made about 14 ft. long and with a uniform width of about 18 in.; but this gave way to the Victorian, Jenny Lind, or broad-tom, usually about 6 or 7 ft. long, 12 in. wide at the upper end, and 3 ft. at the lower. It really consists of two distinct troughs or boxes placed one above the other, as shown in Fig. 20. A stream of water flows in by the spout A, just over the place where the dirt is introduced into the upper box or "tom"

FIG. 20.



GOLD-WASHING TOM.

proper B. The dirt is constantly thrown in by one man, while a second is occupied in stirring it about with a square-mouthed shovel, or a fork with several blunt prongs, which is more useful for pitching out the heavy boulders that sometimes occur, and for tossing back undissolved lumps of clay against the current. To save wear and tear, the floor of the tom is lined with $\frac{1}{8}$ -in. sheet iron. The lower end of the tom is cut off obliquely, so that the mouth may be stopped by a sheet of perforated iron, such as forms the bottom of the cradle-riddle already described. The apparatus being placed on an incline amounting generally to about 12 in., the materials all gravitate with the water towards this sloping grating at the mouth, everything passing through it save the large stones, which gather on the grating and are removed as often as necessary. Beneath this grating *b*, stands the riffle-box C, into which all the fine matters, including the gold, descend. The riffle-box, like the tom proper, is made of rough plank, and is also placed on an incline, but only just so

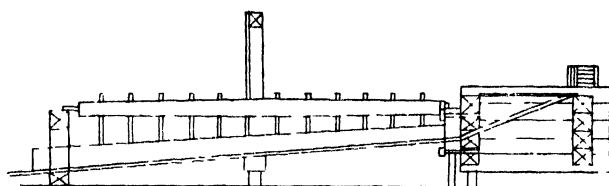
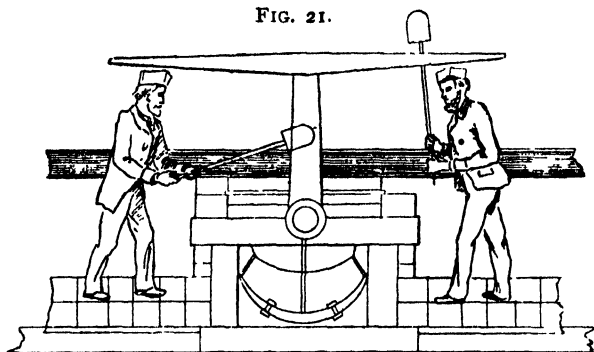
that the water passing over it will allow of the bottom becoming and remaining covered with a thin coating of fine mud. In this way, the gold and a few of the heaviest minerals will find their way to the bottom and rest there, especially by the help of the riffle-bars *c*, which give their name to the apparatus. Sometimes a little mercury is put behind the riffles, so as to assist in retaining the gold, and occasionally the riffle-box is supplemented by a series of blankets, which are useful for catching the very fine gold. The Californian tom, which is the one shown above, differs from the Victorian only in proportions, being generally about 12 ft. long, 20 in. wide, at the upper end, and widened gradually to 30 in. at the mouth. Toms are supported on stones, logs, or trestles, as occasion may demand. They are cleaned up periodically, and the gold and amalgam are panned out as with cradles. They employ 2 to 4 men, according to the character of the dirt and the supply of water. They are applicable to diggings where the gold is coarse; but, though generally free from the drawback of the cradle—the caking of the sand—they are quite incapable of saving all the fine gold.

In Dutch Guiana, according to J. Jewell, the long-tom used is 10 ft. long and 2 ft. wide inside (built of $1\frac{1}{2}$ -in. plank), to which is attached a contrivance called a "torpedo." A false bottom is put on the floor of the long-tom, consisting of $\frac{1}{2}$ -in. plank, which is removed each time a clean-up is made. The torpedo is a sloping (upwards) iron plate 4 ft. long, of the same width as the floor of the tom where it is joined to the latter, but narrowing considerably towards the end where it meets a riffle. The plate is perforated with $\frac{1}{2}$ -in. holes placed $1\frac{1}{2}$ in. from centre to centre, the plate is $\frac{1}{4}$ in. thick. There is a false bottom similarly perforated in the torpedo, and another in the launder leading away from the riffle, and measuring 10 to 14 ft. long. If the torpedo is properly arranged, no gold should be found beyond the riffle.

The apparatus employed by the Siberian gold-seekers for washing auriferous sands is of several forms, all allied to the "tom." In operating that shown in Fig. 21, described by Warrington Smyth, 8 or 10 men may be employed. The sieve or grating, firmly fixed, is $3\frac{1}{2}$ ft. sq. The tye, into which the fine sand is carried by an inclined plane, is 16 ft. long by $2\frac{1}{2}$ ft. wide, with an inclination varying according to the more or less loose nature of the sand, from 24 in. to 16 in., and having its bottom curved in the arc of a circle. Above the tye, and placed in the direction of its length, is a wooden axis turning in gudgeons, and having fixed to it on the under side as many arms as the tye has compartments, each of which arms is fitted with a number of claws or blunt knives; and by an alternating motion given to the axis by 2 men holding a line attached to it, the claws pass backward and forward in an arc, and keep the sediment in continual agitation. The partitions in the bottom are ribs 2 in. high

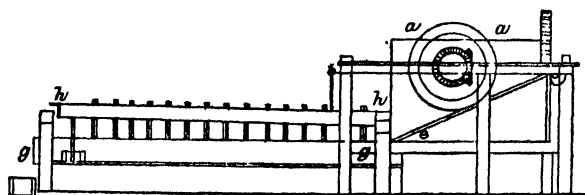
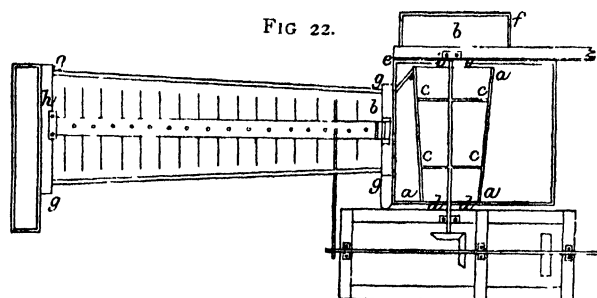
and $2\frac{1}{2}$ in. wide, held by screw-bolts and nuts ; and are removed every now and then for washing out the concentrated auriferous sand, the more frequently as the sands are the richer in gold.

FIG. 21.



RUSSIAN GOLD-WASHING APPARATUS.

FIG. 22.



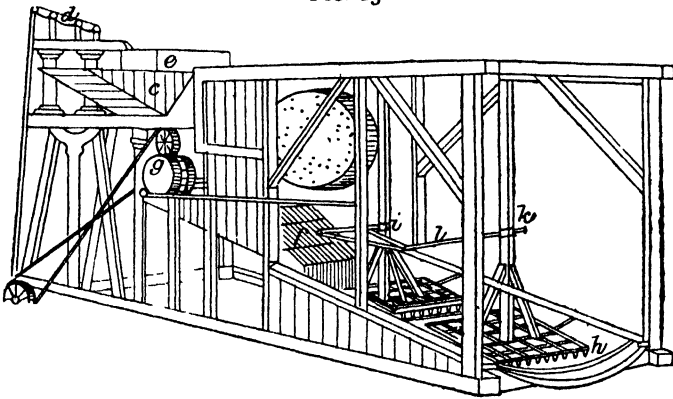
AGTÉ'S GOLD-WASHING APPARATUS.

Agté's machine, used at Alexandrof, is shown in Fig. 22 : *a*, truncated cone of perforated sheet iron, fixed by an iron frame *c* to horizontal

axis *b*, which is made to revolve by a water-wheel about 30 times in a minute; *d*, opening at which the cylinder is charged; *i*, opening at which large pebbles roll out over the board *f*; *e*, inclined plane by which the sand that passes through the sieve is carried down to the tye or long trough *g*, the plan of which is a truncated cone, and whose bottom is concave; *h*, axis above the tye, to which are affixed arms carrying iron knives to stir the sediment, and making 20 to 25 oscillations per minute.

Phillips describes a similar machine for horse-power, used in the Urals, and shown in Fig. 23: *a*, sheet-iron cylinder pierced with $\frac{1}{4}$ -in. holes, and

FIG. 23.



URAL GOLD-WASHING APPARATUS.

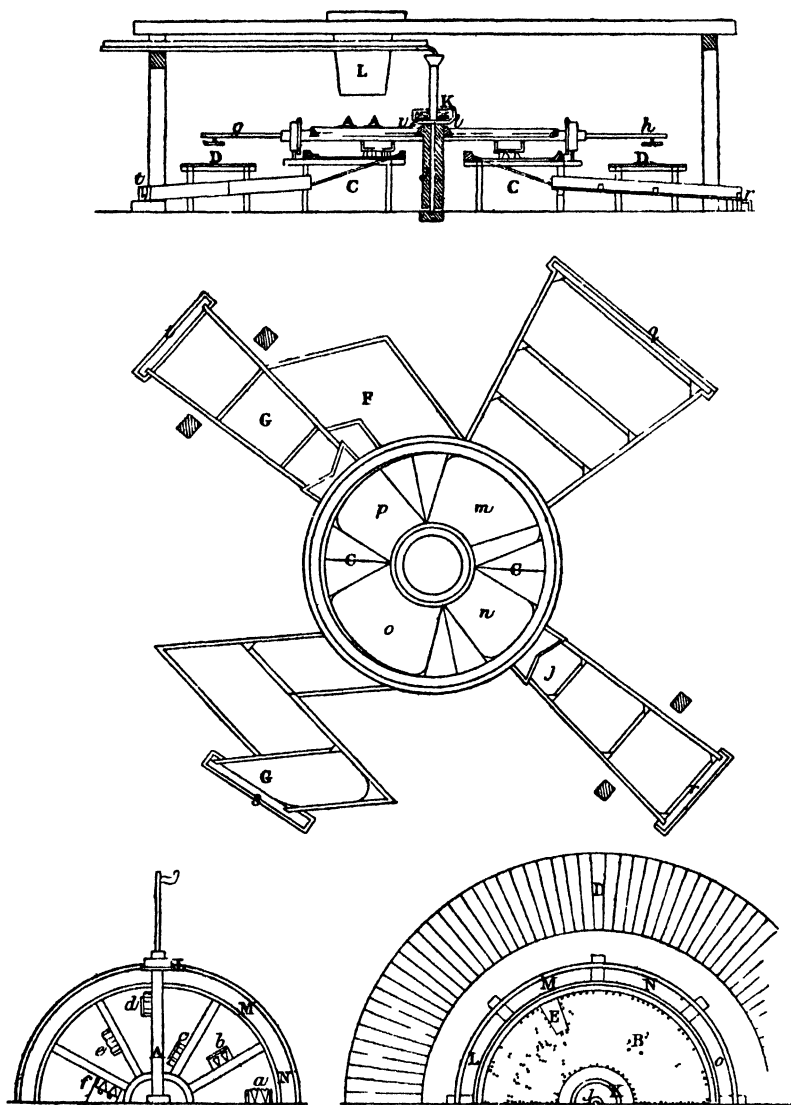
strengthened on the inside by an iron trellis. The cone, which may be about 8 ft. in length, and has a mean diameter of 3 ft. 6 in., is larger at one extremity than at the other, and is fixed on a spindle capable of being set in rapid motion by means of a train of wheels worked by a horse-whim. The auriferous sands are brought in waggons running on iron rails to the hopper *c*, whence they fall into the movable cylinder through the aperture formed by its smaller circumference. At the back of the arrangement is a double pump *d*, set in motion by a crank on the shaft, which communicates motion from the whim to the cylinder *a*. This pump raises water into the cistern *e*, whence it is conducted by 4 iron pipes into the cylinder *a*. These pipes enter the cavity of the drum through the two open ends, and are so arranged with regard to length as to afford a nearly equal supply of water throughout its whole capacity. When set in motion, the perforated cylinder makes 30 to 35 revolutions a minute, and consequently throws, by its centrifugal action, the water and finer particles of sand and gravel through the numerous perforations which it contains; whilst the pebbles and other fragments which are too large to pass through the holes, are carried off through the larger end of

the cylinder, and then fall into a box (not shown). This receptacle will contain any nuggets which may have been present in the sands ; and as, by passing through the cylinder, they will have been washed perfectly clean, they may now be readily seen and picked out. The sand and water having escaped through the apertures in the drum, fall on the inclined platform *f*, which is provided with numerous horizontal bars, for the purpose of separating the heavier from the lighter portions. From this platform, the current flows on to the concave table *h*, also provided with checks, in the form of wooden bars nailed across at distances of about 3 feet from each other. The sands which have arrived at this part of the table are now kept constantly agitated by the wooden pendulums *i k*, provided at their lower extremities with frames made to suit the concavity of the table, and fitted with flattened teeth, like those used in harrows. These pendulums are made to swing by means of the rods *l*, driven by the crank *g*, and are so arranged as to move constantly in opposite directions. In this way the operation is continued, until a considerable accumulation of rich auriferous sand has taken place in the upper part of the tables, where, being retained by the horizontal slips of wood, it remains ; whilst the lighter matters are carried off by the current of water to the lower end of the table *h*, whence they are either made to pass over a fresh series of tables, or if, as usually is the case, they are found to be sufficiently impoverished, they are allowed to run entirely away.

Another apparatus, described by Capt. Razguildeyew, is shown in Fig. 24. Its description is as follows :—A, upper ring, across which are suspended the iron harrows *a b c d e f* ; it performs a rotatory motion by means of the two lever arms *g h*, on the tramway L M N O. B, grinding floor or sieve ; it is a little concave in the centre, and is formed of plates of cast iron joined together : for greater lightness, it might perhaps be made of wood covered with sheet iron. C, lower stage of the apparatus, on which the disintegrated sand can fall from the sieve ; then the sand, borne by water, falls on the washing-tables *m n o p*, which effect the concentration of the auriferous matters. *q r s t* are reservoirs for the dirty water and sand. From these reservoirs the sands are either (according to their assay) finally thrown away, or re-washed on a table. D, horse-walk. E, opening for exit of pebbles and gravel. F, auxiliary table for finally washing pebbles. It is placed under the sieve to avoid transport ; they should fall of themselves upon the table, and there be carefully washed. The dirty water from this table passes over another one adjacent G, on which finally the minutest particles of gold are deposited. K, general receptacle or box of cast iron fixed to the iron spindle *j*. The water from this reservoir is spread by the iron pipes *i l* arranged so as to wet the sands. This same water passes over the

washing-tables and thence into the settling-reservoirs. L, sludge-channel for the sands. The apparatus employs 6 horses and washes 10,000 to 12,000 *puds* (of 36 lb.) per 24 hours

FIG 24



RUSSIAN GOLD-WASHING APPARATUS.

Tatarinoff gives the following description of the washing operations as conducted throughout Central Asia. A framework of light willow

poles, with a convexity of about 35° to the horizontal plane, takes the place of the usual washing-box. Upon this framework, firmly fixed in the ground, is stretched a blanket or carpet to catch the auriferous alluvium. In order to retain the gold, which has of course a greater specific weight, thick horsehair ropes are sewn along the whole length of the blanket. The frame being now ready, a bag-full of ore-stuff is emptied into it; water is then poured on and stirred with a small stick, whilst stones are flung aside.

When 3 to 4 cwt. of alluvial soil have been treated in this way, the men proceed to collect the sediment adhering to the blanket. For this purpose, the blanket is taken off the frame, folded in several thicknesses, and subjected to another washing in the following way:—A wooden bowl or trough, large enough to contain the folded blanket, is placed in a small bay of the river where the current is slack. The workman then removes the blanket, fold by fold, and washes it, allowing the sediment to collect at the bottom of the trough. Having accomplished this, he takes the bowl in his hands, and moves it rapidly with sudden jerks to right and left, splashing the water and the lighter soil over the sides. From time to time more water is added, and the work is continued for several hours without intermission. As the gold in the bowl gradually detaches itself from the particles of sand mixed with it, the movements of the workman become more and more artistic and careful, his whole object being to leave nothing but pure gold, however small the quantity, in his bowl. When this has been obtained, he dips a finger into the water, and flicks a drop or two into the bowl, pouring them out together with the gold into a separate porcelain cup.

The process at Beriozofsk, in the Ural, as seen by the writer, is carried on in a circular wooden frame or box, some 10 ft. in diameter, with a perforated iron bottom, and open at the top, and in which revolves a heavy 4-handled iron rake, driven by steam-power. Into this box, barrow-loads of ore-stuff from the mine are continually emptied. The clay or sand mixed with water is sifted in this way, the lighter particles being carried away, the heavier dropping through the metal sieve. After a second washing, the gold is finally separated from any remaining adherent particles of soil by hand. For this purpose, skilful and experienced workmen stand on a flight of broad wooden steps, over which flows a thin stream of water, clear as crystal. After manipulating the auriferous sand for a long time, the workman at last produces an insignificant quantity of fine gold-dust in the palm of his hand. In this way, 10,000 *puds* (about 160 tons) of ore-stuff daily pass through the Beriozofsk works. So imperfect, however, is the process, that the director stated that a second or even a third washing, an interval of

2 or 3 years having elapsed, yielded better results than the first.* The percentage of gold at Beriozofsk in 1861 was very small—only 30 gr. in 100 *puds*—whilst in former years as much as 2 and $2\frac{1}{2}$ *zlotniks* ($131\frac{1}{2}$ to $164\frac{1}{2}$ gr.) had been obtained. Across the frame, branches or sticks are set at close intervals; the whole apparatus stands 4 ft. 6 in. high by 1 ft. 2 in. wide.

Sluices.—Every sluice consists essentially of an inclined channel through which flows a stream of water, breaking up the earth which is thrown into it, carrying away the light barren matters, and leaving the gold and heavy minerals. Generally, sluices may be classified as “ground-sluices” and “box-sluices,” the former being cut in the surface of the earth, while the latter are of wood, and raised above the surface, necessitating the lifting of the pay-dirt into them.

Here again we find that the modern miner's sluice has been copied from native forms of prehistoric age.

Ground Sluicing.—Besides washing for the gold in the ordinary wooden dishes, the people of the Jashpur State, India, arrange small water-courses before the rains, so as to catch the soil, in which a large proportion of gold is found; these are cleared out from time to time.

Forbes gives the following description of a local method of sluicing adopted in Peru. The valley is, in the first instance, completely closed up, and the course of the river is stopped by a rude wall or dam of stones, provided with sluices, and having a portion of the wall somewhat lower, in order to carry off any overflow of water, which otherwise might disturb the workings. A longitudinal excavation is then made close up to the one side of the valley, and of such breadth as can be conveniently carried on by the number of hands at disposal; and, in making this, the large boulders and stones, too heavy to be carried off by the rush of water, are piled up to one side, whilst the earth, gravel and clay are merely loosened and flushed off, by the water turned on from the sluices, allowing the force of the stream to carry them down the river. On arriving at the several successive auriferous beds, which are known from previous trials, and which are denoted by the dark bands running horizontally across the excavation, as seen in Fig. 25, more care is taken; but the whole of the auriferous earth is likewise flushed off, and, being so much heavier than the rest, deposits itself at but a little distance from the workings, where it is collected and subjected to repeated washing in a trough until nothing but the gold-dust remains behind.

A similar method prevails in Chili. First a cutting is made, near

* This fact implies, perhaps, not so much imperfection in the process, as the presence of pyrites, which, being oxidised by the exposure during a considerable length of time, gave up the gold on further treatment.

the lower end or side of the auriferous deposit, vertically from the surface down to the bed-rock itself; from this point a trench is cut, of sufficient width and fall to carry off the muddy water, to the nearest point at which the ground may be low enough to afford it a free exit. Then water is conducted from the nearest stream that comes from the

FIG. 25.



MODE OF SLUICING IN PERU.

high lands, so as to fall, as if from a spout, over the inner and deep end of this cutting. The ground is thus broken up and disintegrated, and the clay and other earthy matters are washed away by the current, this effect being assisted by the use of rakes with one broad but pointed tooth, called *almocáfres*. When a convenient quantity of ground has been thus broken up and washed, the cutting is 'cleaned up,' that is, the stones which lie heaped in the end of the cutting, under the stream of water, are collected by hand, and built into rude walls on either side of the excavation, so as to occupy the least space possible. If necessary, the stream of water is partially cut off while this is going on. The remaining deposit in the bottom of the cutting will consist of small gravel, coarse sand (some of which is ferruginous), and gold. The gravel and coarse non-ferruginous sand are separated from the gold by washing by hand in wooden pans (*bateas*), and the ferruginous sand is taken out with a magnet. The gold alone then remains. The operation of washing and breaking up ground is resumed until another cleaning-up day comes. When it becomes necessary, the stream of water is shifted to the right or left, and carried forward so as to commence the attack on a fresh strip of auriferous ground, and so on until the whole auriferous deposit be worked out. Recent attempts to replace this system by

hydraulicizing have proved that the older method was cheaper and more effective under the existing conditions.

In Japan, a suitable place having been found, the bank is cleared of grass, roots, and soil for a width of about 3 ft. and from the bed-rock to the top of the bank, exposing the whole thickness of the gravel bed. In the meantime the gold-washers form a washing-ditch by cleaning the bed of the stream for a width of 2 to 3 ft. and a length of 15 to 20 ft. When the stream is large, it is divided, only the volume of water requisite for washing being admitted to the ditch. In thus dividing a stream, care is taken that the stones of the partition-wall are so placed that all leakage of water shall be *from* the main stream into the ditch, and that there may be no flow in the opposite direction, otherwise much of the gravel with its gold will be lost. These preparations completed, the gravel is shovelled into the head of the washing-ditch. At the same time the men wash and throw out the large stones by hand, and separate the smaller stones from the gravel with the *joren* (scoop-shaped baskets) and *kua* (hoes). The rapid current washes out all the clay and fine sand, leaving only the fine gravel in the stream. Riffles are built of flat stones across the lower end of the ditch, to arrest the gravel and to prevent any gold being carried down stream. When a quantity of gravel has been shovelled into the water, forming a bed of fine gravel about 1 ft. thick, washing on specially made straw mats begins. Two or three mats, each 1 ft. wide by 2 ft. long, slightly narrower at the lower ends, are placed side by side across the stream about 2 ft. below the upper end of the gravel. The upper edges of these mats are buried a little in the gravel and held in position by the foot of the gold-washer, usually one man to each mat, the number of men being regulated by the width of the stream. These men now carefully hoe the gravel to the head of the mats, so that it shall be carried over their surface by the force of the current. The heavy gold and iron-sand sink between the thick twisted strands of straw forming the mat, while the lighter gravel passes down-stream. The lower ends of the mats are lifted from time to time and folded over the upper portion, to transfer the gold to the head of the mat and to keep the lower part clean. When all the gravel in front of the mats has been hoed over their surface, the men move them about 2 ft. down stream, and begin to work in a similar manner on the new portion of gravel thus exposed. Finally, after 25 to 30 minutes' work, the mats are removed one by one from the bottom of the stream, folded in both directions under water to transfer the material to the middle, and then bent into the shape of a trough and "jugged" with a longitudinal motion under the water to separate the lighter sand and gravel which still remains. One of the mats is then held in the stream, between the ankles of the gold-washer, and folded lengthwise as a

trough, through which the water is allowed to flow quietly. A second mat is then seized by the ends, inverted and folded transversely, the fold being allowed to dip just below the water in the trough of the mat below. The mats are so woven that this folding opens the crevices between the large transverse strands, allowing the gold and iron-sand to be washed out by the water. By alternately raising each end of the mat while depressing the other, the whole surface is successively washed by the stream, and the various strands of the mat being opened and loosened by the folding, the gold and iron-sand are very completely transferred to the lower mat. In a similar way the concentrated material from all the mats is collected on one, and from this it is finally transferred to a shallow wooden washing-bowl, which is slightly warped and dished out. This last transfer is made with great care in a quiet pool of water, in which place the final concentration on the bowl or board is effected. The separation of the gold from the iron-sand on this board requires great skill and caution. The board is first floated on the water, and by a few oscillations, the material is washed to the centre. Then it is raised from the water with a number of smart longitudinal jerks, the effect of which is to bring the heavier material to the end next the gold-washer, while the lighter sand flows off gradually with the water at the lower end. The board is now brought to the surface of the water, the lower end dipping a little below, and with one or two gentle swings, it is again covered with water, while a small quantity of the light sand is washed from the lower end. The board is then raised with the same dexterous shakes as before, the washing being conducted with great care, and but little iron-sand being allowed to pass off each time. After 5 or 6 minutes, most of the gold will have been brought to the head of the board, the light-yellow grains showing very distinctly in the black sand. The gold may be brought more clearly into view by allowing a little water to trickle down the inclined board, which washes off the black sand and leaves the gold exposed. The first portion of gold, with some of the adjoining iron-sand, being removed, the remaining sand is washed a few times, usually yielding one or two grains in addition. Finally, the gold and the small quantity of iron-sand taken from the board at the same time, having been dried, is submitted to a careful separation by blowing, the iron-sand is carried off, and the gold remains ready for weighing. In this last separation, a small magnet is sometimes of assistance for removing the large grains of iron-sand, too heavy to be blown away.

Pliny gives a detailed account of the sluicing operations carried on in Spain in his day, where the undertakings were on a scale rivalling the great alluvial claims of California. The sluicing trenches were lined with planks at the sides, and covered at bottom with a layer of plants of the genus *Ulex*, whose rough and prickly foliage served to arrest the gold.

When the amount of water available is not constantly sufficient for a box-sluiice, but a "head" can only be got for a short period, immediately subsequent to heavy rains, it would not pay to erect a box-sluiice; but recourse may then be had to a ground-sluiice, provided there is an abundant fall and outlet for the tailings. Ground sluicing also requires about six times as much water as box-sluiicing to do the same amount of work. The ground-sluiice consists simply of a gutter, formed partly by taking the stream through it, and assisted by loosening the earth with a pick. When the gutter has reached suitable dimensions, the pay-dirt is either washed in by the stream, or conveyed thither by manual labour. If the bottom be a hard and uneven bed-rock, its inequalities will suffice to arrest the gold; if not, a number of boulders too heavy to be moved by the stream must be thrown carelessly into the sluiice. No mercury is used, no planking, and no riffles. It is never intended for continuous working, nor will it save any but the coarse gold. In order to clean up, the water is diverted from the channel, and the auriferous matters are collected, to be panned out, or washed through a cradle, tom, or short box-sluiice.

Box-sluiices.—The box-sluiice or board-sluiice, as it is also called, is a long wooden trough or series of troughs, and is now probably the most important implement used in placer mining. Its length varies from 50 ft. to several hundred or even several thousand feet, its width is never less than 1 ft. nor more than 5 ft., being generally 16 to 18 in., and the height of the sides varies from 8 in. to 2 ft. For convenience and speed in construction and removal, the sluiice is made in sections or lengths of about 12 or 14 ft., each such length or section being called a "box." It is built of 1½-in. rough planks, the bottom boards being sawn 4 in. wider at one end than at the other, so that the narrow end of one box telescopes into the broad end of the next throughout the whole series, and beyond this no nailing nor fixing of any kind is required. This line of troughs rests on trestles, and is so arranged that there is a descent throughout the whole series. The amount of this descent, incline, or "grade" ranges from 8 to 18 in. per 12 ft. A fall of 8 in. in 12 ft. is called an "eight-inch grade," and so on. Generally the grade is uniform throughout, but that is not invariably the case. Many points have to be considered in deciding the grade of a sluiice, and it is inadmissible to have less than 8 or more than 20 in. fall in each 12-ft. length. It is important that the sluiice should be conveniently near the level of the ground at the point where the pay-dirt is introduced, and this will naturally have a bearing upon the grade, as will also the character of the pay-dirt and the length of the sluiice. The steeper the grade the more quickly the dirt is disintegrated, but the more likely is the fine gold to be washed away by the force of the water. The tougher the dirt the steeper must the grade be;

tough clay, for instance, does not dissolve so quickly in a slow current as in a rapid one. The shorter the sluice the smaller the grade should be, as there is more danger of the fine gold being lost in a short sluice than in a long one. The steeper the grade, the greater amount of work it can do, and the greater the loss of fine gold. In Australia, some years since, no miner was satisfied with less fall than 3 in. in every 12 ft., but now many claims are worked with a fall of only $1\frac{1}{4}$ in. But obviously much depends on the nature of the wash-dirt. To give good results there should be a full supply of water, and generally the depth of water on the floor of the sluice should not be less than 2 in. As ordinary pay-dirt is generally completely dissolved in the first 200 ft. of a moderately low-grade sluice, the extra length is useful only for catching the gold; sometimes therefore the grade of the last part of the sluice is reduced. Occasionally it happens that the clay is so tough that it will roll for $\frac{1}{4}$ mile in large balls, not only refusing to break up, but also doing much mischief by picking up the gold in its passage. Such clay must not be put through a sluice, but submitted to a puddling process, to be described hereafter.

The sluice-box used in Dutch Guiana is thus described by J. Jewell. A large box on the line of the sluice receives the auriferous alluvium; here a man stands with a wooden rake to keep the clayey and sandy matters in motion. The sluice is made in 10-ft. or 12-ft. sections, about 1 ft. wide, and with a total length of 80 to 100 ft. if possible. Riffles are placed at each junction. At the third length of the sluice stands a boy whose duty it is to catch all lumps of clay and transfer them to a small puddling-box, 5 ft. $\times$ 3 ft., which is joined to the side of the first length of the sluice; here they are thoroughly broken down by means of water brought in from the main supply, and are finally let out as soft mud into the first length of the sluice, to be washed again like the other material.

In Borneo the auriferous earth is washed "in an aqueduct lined with the bark of trees, and supplied by a neighbouring brook with a stream of running water. This is nothing more nor less than a box-sluice.

Nearly all the gold-working in Borneo is done by *kunsis* or companies, which sometimes number several thousand men, generally, however, only a few hundred. St. John thus describes the mode of working by the Chinese at Bau. They dam up the end of the valley, forming a large reservoir of water $\frac{1}{4}$ mile long. The dam is completely faced with wood, towards the water, and partially on the outside, to resist the tropical rains. A ditch 4 ft. wide is cut from the reservoir to the ground selected to be worked; a sluice-gate in the dam regulates the supply of water to the main ditch, and minor sluice-gates to it regulate the supply to the smaller ditches. When this is all prepared, the sluice-gates are opened, and the earth in its neighbourhood is thrown into the ditch; the rushing

water carries off the mud and sand, and allows the particles of gold to sink to the bottom. After 3 or 4 months, they clean out the ditch, and carefully wash the residue, which generally yields them sufficient to make a tolerable division among the workmen after all the expenses are paid.

Following is Muniram's account of gold-washing in Assam, where the same principles are put in practice :—

Wherever the current is strong, with a falling bank above it, ending in a sharp turn of the river, the Sonwals examine the opposite shore where the sand from the falling bank is thrown, and if this should contain gravel mixed with the sand, it is accounted a good place to find gold in. Each party consists of a headman (*patoee*) and 4 assistants (*pallees*), who wash in one trough (*dorongee*). When they find a proper place to commence operations, they begin by working about in the sand with a sharp-pointed bamboo (*sokalee*), to find the depth at which the gravelly sand is ; they then take it up in a piece of split bamboo (*bans chola*) and examine whether there is any gold-dust in it. If they see 12 or 14 bits, they immediately build their houses and commence operations. They first bund up the deep part of the stream, if it be a small one, with sand, and if large, with stakes and grass : the stream then takes a different direction over the sand ; they allow it to wash away the upper surface of sand, so as to expose the gold sand, when the bund is re-opened, and the stream returns to its original bed. The upper sand is then scraped off, and the gold sand is collected with a kind of wooden spade (*kater dohtal*). This shovel is $1\frac{1}{2}$ cubits long by 1 cubit in breadth, with a handle 4 cubits long ; the blade is of the form of a crescent, with holes at each corner, through which a string is passed, and 2 men lay hold of and pull this string, while another keeps pressing the spade down in a perpendicular position. The sand is then taken up in small baskets with handles (*cookees*), and thrown on a bamboo latticework or strainer (*ban*), which is laid over the trough. This trough is made of wood, and is 3 cubits long, 1 cubit broad, and 1 span high all round, with a slit 3 fingers wide at one end. Water is now thrown over the sand with a calabash having a large piece scooped out at the bottom, beside a very small hole on one side (*lao*) ; the water is thrown on with one hand while the other hand is employed in moving the sand about, and sweeping off the larger particles of gravel from the surface of the strainer. In this way, the sand is spread on, and water is poured over it ; and as the trough fills, the water and dirty sand run off through the slit in it, while the clean sand and gold remain at the bottom of the trough, the trough being placed at a small angle to assist the water and dirt to run off quickly. When 40 or 50 baskets of sand have been thus washed into the trough, the Sonwals call it a *shia*, and if a *rati* of gold is produced from

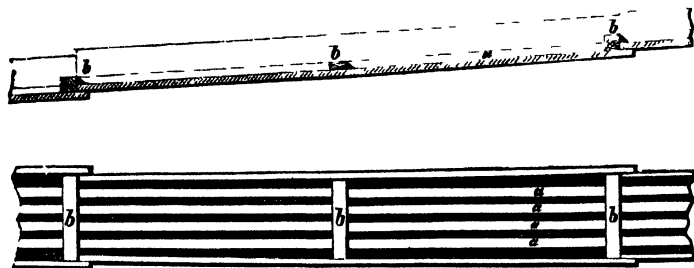
one *shia*, they think themselves very fortunate indeed ; for during the long days they get about 30 *shias* or washings producing 1 *rati* each, and during the short days about 25 *shias*, each party thus making on an average about $\frac{1}{4}$ *tola* (45 gr.) of gold daily. When they happen to fall on a good old stream that has not been disturbed for 5 or 6 years, they get 2 *ratis* of gold from every *shia* or washing, and then each party makes about $\frac{1}{2}$ *tola* (90 gr.) daily. The gold and sand of the last washing is collected into pottles (*chonga*), by spreading a leaf of *copat* or some other plant at the end of the trough, and dropping water very gently on the sand through the small hole in the calabash, which causes a parting of the sand and gold thrown on the leaf ; when the whole is collected in this way, it is put into the pottle and tied up, and the next washing is commenced. As soon as they have collected enough in the pottles, they give up washing the common sand, but pour out the gold and sand from the pottles into the trough again, and putting in about an anna's weight of quicksilver for each *tola* (180 gr.) of gold-dust, they pour water over the sand to keep it in motion, while the quicksilver remains below with the gold-dust and forms it into a lump ; this lump is then put into a shell on a fire of *nahar*-wood charcoal. When the quicksilver evaporates the shell becomes lime ; it is then carefully taken up in a spoon, and thrown into water, when the gold falls to the bottom. If it be of a brass colour, it is wrapped in a paste made of clay from the cooking *chulas*, mixed with a little salt, and burnt in a fire, which gives it a proper colour. This process causes an absolute refinement of the surface of the gold. It is the same used in gold refining by the natives generally ; but in the latter case, the metal has to be reduced in the first instance to very thin leaves, to allow the muriatic acid fumes to penetrate and unite with the alloy.

False bottoms and riffles.—When the grade of a sluice is very low—say 1 in 40 or 50—the gold is easily caught, and much of it would rest even on the comparatively smooth floor of the sluice ; but additional means of catching it are always adopted. When stones are plentiful in the wash-dirt, a small bar may be placed across the lower end of each trough, to prevent the bottom from being run bare. From such sluices, it is usual to throw out all stones as large as a doubled fist, for which purpose a fork with several blunt prongs, called a “sluice-fork” is used. The same serves to loosen occasionally the materials which have collected behind the bars, and in all sluices watchfulness has to be exercised that the boxes do not choke, and thus send the contents over the sides to be lost.

Some kind of false bottom is almost always used in the sluice, destined to catch the gold and save the wear and tear on the floor of the sluice itself. In Victoria, where the water supply is limited and the

sluices are of moderate scope—often only 25 to 60 ft.—boards bored with as many 1-in. or $1\frac{1}{2}$ -in. holes as there is room for are fitted into the troughs. In California and elsewhere, the false bottoms are generally composed of longitudinal riffle-bars, about 6 ft. long, 3 to 7 in. wide, 2 to 4 in. thick, two sets being required for each length of the trough or sluice. Fig. 26 shows the mode of arranging the riffle-bars *a* in the sluice. They

FIG. 26.



BOX SLUICE, SHOWING FAISE BOTTOM.

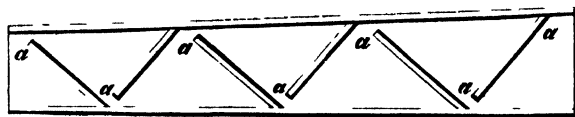
are kept in place by cross wedges *b*, at a distance of 1 to 2 in. apart, and are not nailed, as they have to be removed at every cleaning up. Into the spaces thus formed, the gold and other heavy bodies will fall, always sinking through the lighter particles to the bottom. When the sluice-boxes are no longer fit to be used, or for any reason are not to be used any more, they are dried and burned, and by very carefully washing the ashes, enough gold is often got to buy a new set of boxes.

Block and zigzag riffles.—Where there is a great quantity of pebbles and boulders in the pay-dirt, the riffles just described are worn away very rapidly, sometimes in a week or less. It is then advisable to use "block" riffles which, though more expensive, last three times as long, and therefore save much labour. Instead of being sawn longitudinally with the grain of the wood, they are cut across the grain, so that the fibres stand upright in the sluice-box as in the living tree. But it is difficult to procure these riffle-bars more than 3 ft. long, and they are therefore fixed transversely about 2 in. apart.

The arrangement of zigzag riffles is illustrated by Fig. 27. The first bar is nailed to the bottom at an angle of 45° to the side of the sluice, and reaches diagonally across to within an inch or so of the other side. Immediately below this space *a*, another bar is fixed at a right angle to the first, and reaching from that side of the sluice-box to within an inch or so of the other side, and so on alternately for a considerable distance. The sluice always terminates, however, with a certain stretch of the ordinary longitudinal riffles, as the gold and heavy matters

are not completely caught by the zigzag riffles, but only caused to follow a tortuous course, while the lighter materials are swept right over them.

FIG. 27.



ZIGZAG RIFFLES.

Howland's riffle.—Howland's riffle, made by Morey and Sperry, 92, Liberty St., New York, is made in sections, of cast iron. Each section forms an anticlinal ridge, and is corrugated with parallel channels running up and down the inclined sides, and contains a number of troughs for holding mercury. The eddies produced by these riffles are claimed to present great facility for the valuable matters to exercise their superior specific gravity, and sink into the grooves, while there is considerable exposure of mercury, with which the gold is thoroughly brought into contact.

Amalgamation.—Though the heaviest particles of gold would be effectually caught by the contrivances mentioned, an immense amount of fine gold would undoubtedly escape, but for some additional safeguard. One of the means best adapted to this end is to form an amalgam by introducing mercury to the presence of the gold. This is done in several different ways. When using zigzag riffles, a vessel containing mercury, and pierced by a small hole which allows the metal to escape in minute portions, is placed near the head of the sluice. Trickling down from riffle to riffle, it overtakes the gold, absorbs, and retains it, the amalgam thus formed being caught in the longitudinal riffles farther down. In the ordinary longitudinal riffle-sluice, some mercury is poured in at the head of the sluice about $1\frac{1}{2}$ or 2 hours after starting the washing, and gradually finds its way down with the current, but remains principally in the upper boxes. Smaller quantities are introduced at intervals lower down, the quantity being increased in direct proportion to the amount of fine gold present. Another plan consists in impregnating the pores of the wood forming the riffle-bars with mercury. This is effected by driving a piece of gas-piping ground thin at one end into the wood, and filling it with mercury: the pressure of the column forces a certain amount into the fibres of the wood. This catches the gold, and the resulting amalgam only needs to be scraped off the surface of the wood. The method is rarely used, however, and not generally recommended.

Copper plates.—A fourth plan, of whose efficacy very favourable

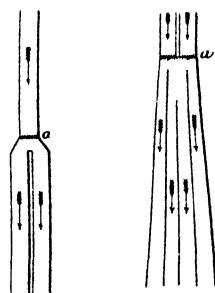
accounts have been given, especially when there is much exceedingly fine gold in the wash-dirt, is the amalgamated copper plate. It is considered as effective for saving fine gold as an equal surface of mercury, being at the same time cheaper and more easily managed. But it is not nearly so much used in sluicing gravels as in connection with stamp batteries for veinstuff. It measures generally 3 ft. wide and 6 ft. long, varying in accordance with the capacity of the sluice. Sometimes the stream is split and carried over 2 or 3 separate plates. It is placed nearly level, and at a considerable distance from the head of the sluice, as it is intended only for catching the very fine or "float" gold, and for this reason also a sheet-iron screen perforated with holes $\frac{1}{2}$ in. $\times$ $\frac{1}{16}$ in. is placed in front of it, so that only the finest particles pass over it. It is amalgamated by first washing over its upper surface with weak nitric acid, and then applying some mercury, which has been treated with dilute nitric acid, to form a little nitrate of mercury. If the operation is once effectually performed, it will never need repetition, some fresh mercury only being dropped on it as fast as the gold converts it into amalgam. Two points essential to success are that the current be slow and shallow, so that every particle of gold may come into contact with the face of the plate, which is the principal reason for dividing the stream when the sluice is a large one. A freshly-amalgamated plate is liable to become coated with a green slime due to the formation of subsalts of copper, and is then incapable of absorbing the gold. This slime must be carefully scraped off, and the place where it has been must be rubbed with fresh mercury. It has been observed in practice that when a grain of gold has attached itself to the surface of a copper plate, other particles gather around it, evincing a marked preference for those portions to which gold has already adhered over those which are still free from it; therefore the amount of amalgam allowed to collect without being cleaned off will depend chiefly on the means that can be adopted to prevent its being stolen. To remove the amalgam, the plate is taken up and heated to such a degree that the hand cannot be borne on it, except momentarily; this suffices to render the amalgam soft and loose, and it may then be easily scraped off. The plate is allowed to cool, and is again rubbed with a little ordinary mercury, to make it ready for further use. The plate employed should not be less than $\frac{1}{16}$ in., and must be handled with great care, as the effect of the mercury is to render it as brittle as glass. Its surface should be kept clean while in use by constant rubbing with a rubber squeegee.

Cleaning up.—The cleaning up of the gold, mercury, and amalgam caught in a sluice is usually effected after every 6 or 7 days' run, though it is sometimes protracted to every 10 days, or even till the riffle-bars require replacing, the delay entailed by it being avoided as long as

possible. Affording a light half day's work, it is commonly reserved for Sundays. The first step necessary is to cease feeding in the pay dirt and to let the water run through till it issues in a clean stream. Commencing then at the head of the sluice, 5 or 6 sets of riffle-bars are removed, and the dirt dislodged is washed down while the valuable metals lodge against the first remaining set of riffles, and are scraped out with a spoon and placed in a receptacle. The next half-dozen sets are treated in the same way, and thus the process continues to the end. Sometimes the riffles are removed more rapidly by taking up all save one at each 30 to 40 ft. The amalgam and mercury taken out must be pressed through buckskin or canvas to remove the excess of mercury, which will run into a vessel placed to catch it. The remaining spongelike mass of amalgam must be retorted to extract the gold.

Multiple sluices.—Among the modifications of box-slucies may be mentioned the double sluice. In this, the boxes or troughs are made of double the usual width, and divided by a longitudinal partition. They enable one section to be kept at work while the other is being cleaned up, and are useful where the supply of water is at some seasons sufficient for both channels and at others only for one. A modification of this, successfully adopted for the purpose of saving fine gold, is illustrated in Fig. 28. The plan consists in dividing the sluice at, say, the

FIG. 28.



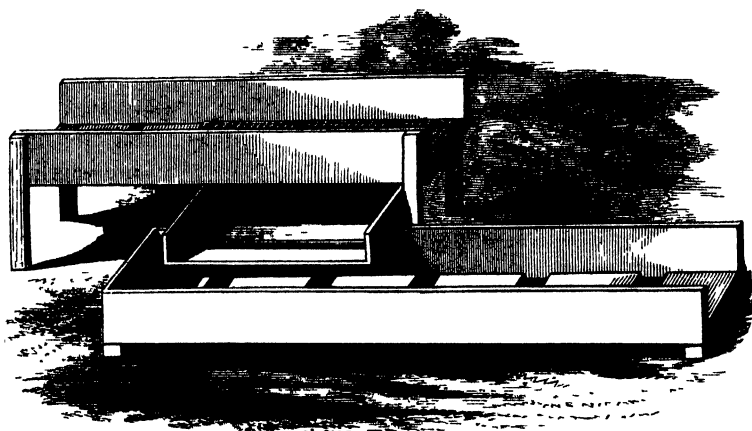
MULTIPLE SLUICES.

centre of its length, shifting the lower part on one side a distance of half its width, and placing another sluice of the same size beside it, so that the two "tail-slucies," as they may be called, cover equal parts of the lower end of the sluice. At the end of the single or head sluice, a grating *a* should be fixed, to prevent coarse gravel washing into the double or tail-slucies. The object is the division of the stream of water when it leaves the head-sluice into two equal parts, and spreading it over double the area of bottom board, so as to decrease its force, and thus lead to the saving of much of the fine and light gold, which is lost by the ordinary sluice. A further elaboration of the same idea is known as the "fantail-sluice" (see Fig. 28), wherein the tail-sluice is again sub-divided, a second grating *a* being placed at the junction, as before.

Under-current sluice.—An important apparatus, known as the "under-current" sluice, is represented in Fig. 29. The bottom of the last box of the sluice is partly covered with an iron grating, whilst underneath the grating another sluice is introduced, furnished with a fresh supply of water, and set at a lower grade. The end of the last box of the upper

sluice being left open, all the large stones which will not pass readily through the grating are carried by their own impetus out of the sluice, and a pit or other accommodation must be provided for their disposal. The fine matters, together with a portion of the water, will fall through

FIG. 29.



UNDER-CURRENT SLUICE.

the grating into the under-sluice, where a low grade and consequently slow current causes the arrest of much fine gold that otherwise would have escaped.

Evans and Frey's sluice.—The sluice invented by Evans, and improved by Dr. Frey, of Sacramento, is made of cast iron, with transverse corrugations on the bottom, semicircular in shape, and 3 in. deep. At the bottom of each alternate corrugation is a narrow slit, through which the heavier material falls down into another riffle below, with larger corrugations. Both riffles are set on the same grade, which should be about 1 ft. in 12. The lower box is charged with mercury. These sluices are said not to cake, nor to require any attention beyond cleaning up once a week, while they are reputed to have stood the test of use below all other contrivances for saving gold.

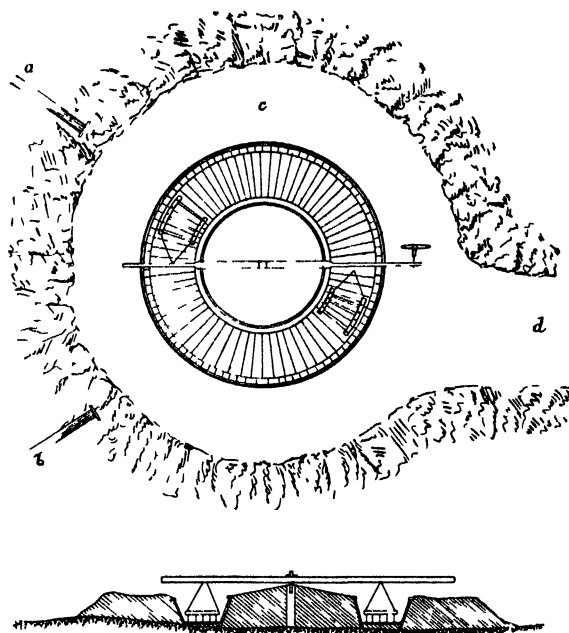
Working results.—The amount of earth which can be washed through a box-sluice in a day per man employed will be limited by the capability of the men. When the height of the lift is $4\frac{1}{2}$ to 5 ft., and 4 men are employed, 24 cub. yd. will be a good day's work, or say 6 yd. per man. With free earth, the tom may do as much; but it cannot exceed that figure, as the men are not able to lift more. In ground-sluicing, on

the other hand, the quantity of water, character of the dirt, and inclination of the sluice, govern the result. Any number of men up to 6 may be employed, but the common numbers are 2 and 3. The quantity washed per man ranges between 20 and 50 yd. daily, the average being probably about 25 yd. per man per diem. Constant watching must be kept up day and night to prevent the sluice being clogged and to guard against theft.

Puddling-machines.—As may be imagined, the machines hitherto described are quite unfitted for dealing with tenacious clay, which often occurs abundantly in alluvial formations, and is frequently highly auriferous. Special implements have therefore been devised for treating such gold-bearing clays, and these are known as puddling-machines. The original and most simple form is the puddling-box,—a rough wooden box about 6 ft. square and 18 in. deep; or even half a large barrel is commonly used, where operations are conducted on a small scale. Into this the clay is thrown, and supplied with plenty of water. The mixture is then stirred constantly with a shovel, prong, or rake, till the clay is thoroughly disintegrated and suspended in the water, when a plug inserted near the bottom of the tub is withdrawn, and the thin mud or rather muddy water is allowed to escape. A fresh supply of clay and water is now admitted, and the stirring process is repeated; this continues until the box has become filled with gravel, sand, &c., to the level of the plug-hole, when operations are suspended, and the deposit containing the gold, if there was any in the material treated, is taken out and washed up in a cradle or pan. Where operations are more extended, an upright shaft furnished with strong arms is placed in the centre of the tub, and set in motion by a mitre-wheel driven by horse- or steam-power. In California, puddling is seldom resorted to, because there is generally a sufficient head of water for hydraulicing, wherein the water force is so great as to break up the clay pretty effectually. In Australia, on the other hand, water is not so plentiful, and there the puddling-machine is in common use. The form is generally that shown in Fig. 30. It consists of a circular space, whose sides and bottom are lined with hard wood or iron, and in the centre of which revolves a perpendicular shaft worked by horse-power. From the cross-beam on this shaft depend two harrows, which are dragged round the circle, and puddle the dirt on their way. Sufficient water must of course be continually supplied. The gold and sandy matters collect at the bottom of the circle, and are removed at intervals and panned out. *a* is the water-supply drain; *b*, discharge drain; *c*, horse-walk; *d*, roadway. Occasionally steam is used in lieu of horse-power. On one occasion, a considerable quantity of silt was taken out of a sludge-drain, and subjected to crushing and amalgamation; it yielded an average of nearly 2 dwt. per ton, which was

considered by competent authorities to represent about the average loss from puddling-machines used at Sandhurst, an indication of the richness of the dirt.

FIG 30

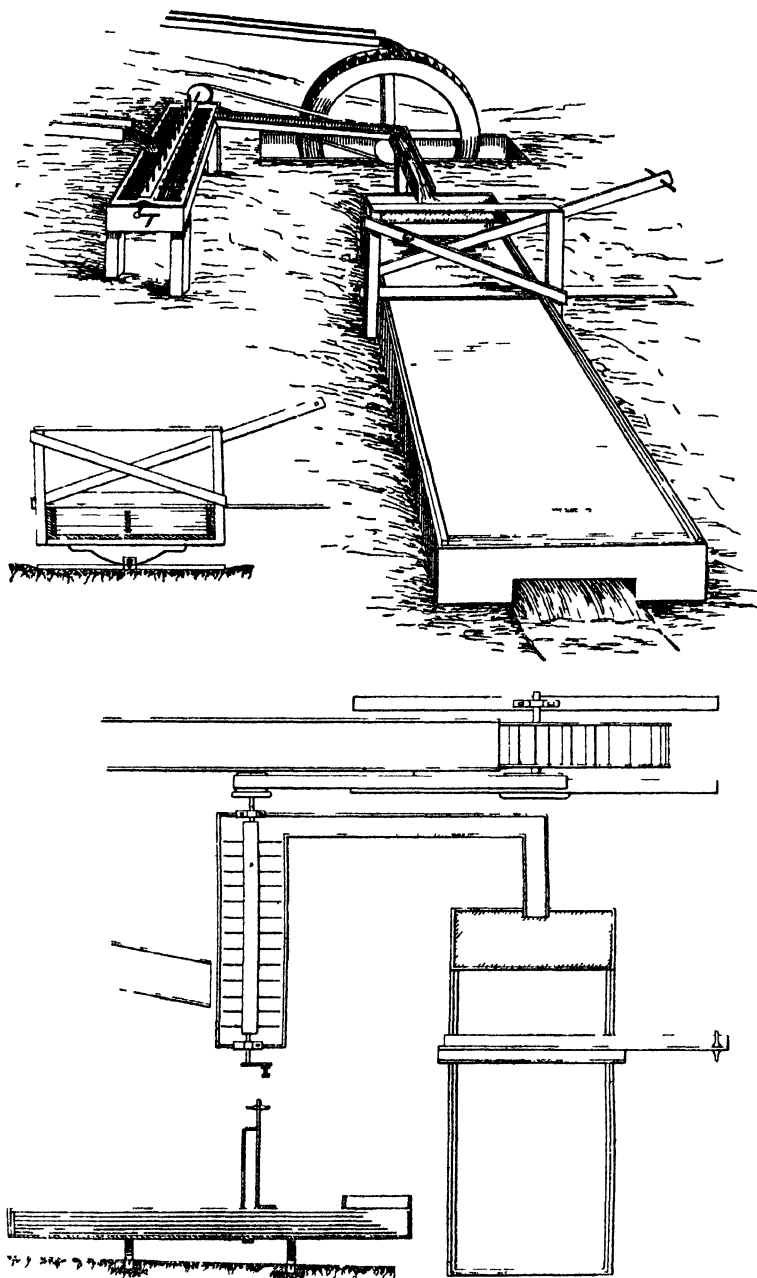


HORSE POWER PUDDLING-MACHINE.

Hart's patent cylinder puddling-machine.—This machine was so constructed that the stones and other refuse could be cleared away without disturbing the gold, which was cleaned up only once a week. For some reason, it never came into general use; but it probably suggested the idea of a cylindrical puddling-machine and sluice combined.

Combined cradle and puddling machine.—Sketch, plan, and sections of this machine are shown in Fig. 31. The necessary power is derived from an overshot water-wheel 8 ft. in diameter, which communicates with the puddling-shaft by two drums and a belt, and with the cradle by means of a rod connected with a crank fixed at the other end of the puddling-shaft. The wash-dirt is thrown into the puddling-trough, which is supplied with water, and in which revolves an axle furnished with 25 iron arms. Hence the wash-dirt mixed with water flows into the hopper at the top end of the cradle. The bottom of this hopper is of sheet iron perforated with $\frac{3}{8}$ -in. holes. In the cradle are 7 tiers of blanket-tables, made of $\frac{1}{4}$ -in. pine boards, and varying in length so that each one receives its due proportion of sludge from the hopper. The

FIG 31

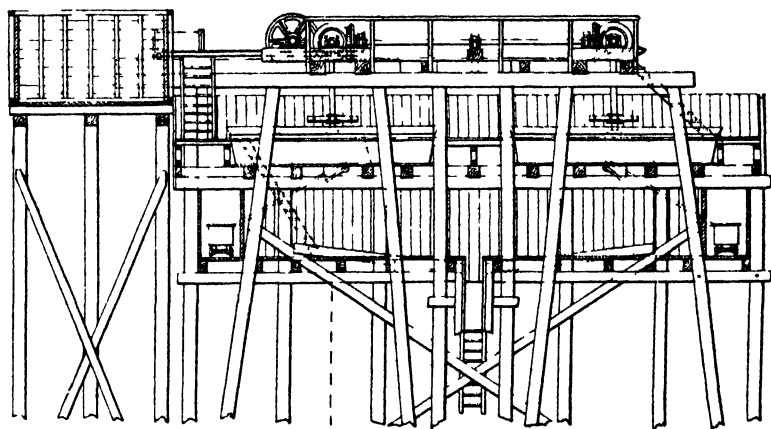


COMBINED CRADLE AND PUDDLING-MACHINE.

blankets are washed 3 times a week, and the slime is concentrated in an ordinary small cradle, the gold being finally amalgamated. The plan and sections are drawn to a scale of 6 ft. = 1 in. The cradle has an inclination of 6 in., and an oscillation of 12 in.

Gibson's Self-emptying Puddling Machine.—These machines are employed by the Berry Consols Company, whose plant is admittedly one of the most complete in Australia, and with general satisfaction. They are illustrated in Fig. 32. The machines are of cast iron, 16 ft. 6 in. diameter by 2 ft. 6 in. deep, in addition to which there is a 9-in. level splash-plate bolted on to the top flange. The side plates and bottom are cast in eight sections, each flanged and bolted

FIG 32.



GIBSON'S SELF ACTING PUDDLING MACHINE.

together with strips of deal to make the joint. The centre drum is cast in one piece, and carries a bridge and foot-block for the vertical shaft which is 7 ft. long by 5 in. diameter, and on which are keyed the arm-plate and crown-wheel. The arm-plate has four ways for wooden arms, to the end of which circular harrows, about 2 cwt. each, are attached. The lying shaft, $4\frac{1}{2}$ in. diameter, is driven directly from the shaft of the engine; the crown-wheel, by a pinion and disengaging clutch. The engine is an 18-in. cylinder, and is steamed from the machinery boilers on the surface. The machines and sluices are served with water from a 10-in. diameter plunger, worked from the crank of a counter-shaft. The machines and engine are erected on a wooden framing, about 43 ft. above the surface. In operating the machines, the débris is run in trucks into the pick-ups, and dropped into the machines; the harrows soon reduce the mass to a puddle; it is then thinned down and allowed

to run away. The quartz boulders are forked off and thrown into the centre of the drum, where they gravitate into the truck, and are then trucked away. The machine being full, the emptying door is opened from the under side of the machine or sluice-plat, and the débris is let down. To assist in this operation, which usually takes about 15 minutes, a semi-circular scraper is attached to the wooden arm, in place of a harrow; this breaks up the dirt and precipitates it through the opening in the bottom of the machine on the sluice-plat, where it is sluiced off. In the usual course of working, one machine reduces about 120 cub. yd. of alluvial dirt as raised from the mines in 24 hours.

Buddles.—These are very rarely used in placer mining, though commonly employed in quartz-mining. They will be described in Chapter VI.

Whips and Whims.—Many contrivances are employed for raising earth from such depths as are beyond the limit of the level whence men can conveniently throw it out with shovels. One of the first means adopted in the earliest days of Australian gold-mining was that of erecting platforms or stages at convenient intervals, and the dirt was thrown from one up to the next by relays of men on each stage. Immediately succeeding this came the wooden windlass, a rude implement made entirely of wood.

Hand-whip.—This was soon followed by the German lever or hand-whip, an arrangement exactly resembling the *chadou*s of Egypt, and represented in Fig. 15, p. 158. It consists of a long, straight sapling, fastened by a rope to a rough post fixed vertically in the ground, or to a living tree if it be sufficiently near. This sapling is made fast at about the centre, and to its thin end is affixed a rope terminating in an iron hook for holding the bucket, while to the thick end is tied a log, or boulder, or bag of sand, sufficient to counterbalance the weight of a bucket of dirt. Two men are required, one filling the buckets below, while a second on the surface hauls them up as fast as they are filled. The weight on the thick end of the sapling enables him to do this with ease.

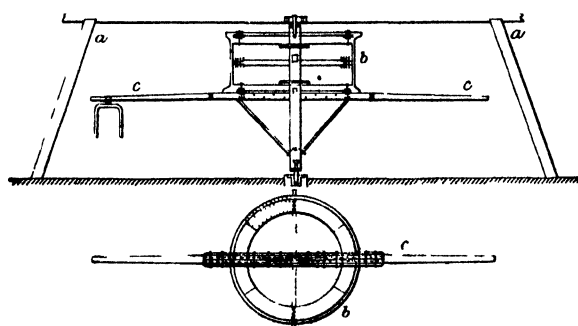
Horse-whip.—Whilst a variety of improvements were being made in the construction of windlasses, the horse-whip was introduced. In the most simple form, it consists of a post about 20 ft. long, fixed at an angle of about 45° to the ground, and firmly imbedded in it. The smaller end of the post should rise about 8 ft. above the centre of the mouth of the shaft. On this end is fixed an iron wheel or sheave, with a grooved circumference in which a rope runs. Another form consists of a set of legs with cross-logs on the top, and a pulley-wheel working on an axle fitted into the cross-logs, similar to the poppet-heads now in use. The rope passes over the pulley-wheel, and is made fast at one end to the harness, while the

other is lowered down the shaft. The horse is then driven from the shaft along a road laid down for the purpose, till the bucket which had been hooked on to the rope at the end that was lowered into the shaft is raised to the proper height. When the bucket has been emptied, the rope is unhooked from the swingle-tree, the horse is turned round, the rope is hooked on to the horse's collar, and, as the horse walks towards the shaft, the bucket is lowered again. For the purpose of emptying the dirt from the bucket, a hook or pair of pincers, attached to a rope made fast to the framework, is inserted into a ring or band at the bottom of the bucket, which is raised nearly to the pulley-wheel for that purpose, and then, as the bucket is lowered by backing the horse, it is drawn aside from over the shaft, and as the rope by which it has been drawn up the shaft is slackened, the rope attached to the bottom of the bucket being tight, the bucket is capsized, and the contents are shot into a barrow or truck placed to receive them; the bucket being emptied, the hooks or pincers are removed from the bottom, and it is lowered into the shaft again.

Water-power derrick.—Raymond mentions a curious water-power derrick, where "the power is transmitted more than 40 ft. by means of a rope from a small overshot wheel to the pulleys of the crane."

Horse-whim —This is represented in plan and section in Fig. 33. A strong framework *a* of timber holds in position a large drum *b* which

FIG. 33



HORSE-WHIM.

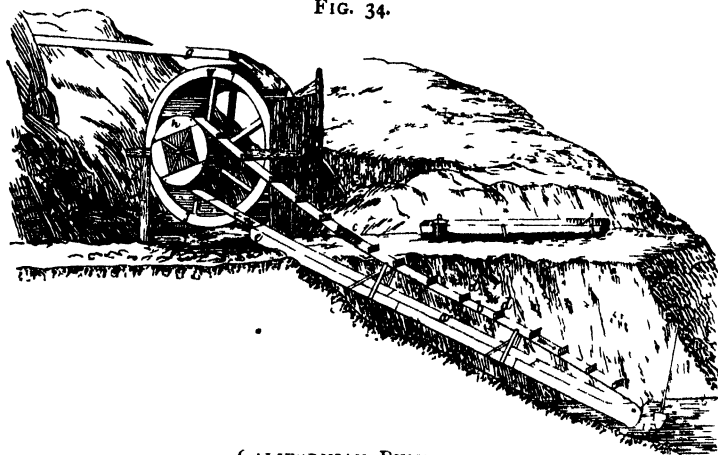
works horizontally, and around which are wound the ropes attached to the buckets working in the shaft. Underneath the drum is a long beam *c*, with shafts to which the horses are harnessed.

Draining the Workings.—The means which first suggested itself for getting rid of the water from alluvial diggings was as rude as the manner of raising the pay-dirt. Buckets, or skins looped up with iron

rings, were either drawn up by hand or by the windlass. Two simple and effective methods have been much used where the configuration of the ground was suitable—the Californian pump, and syphons.

Californian pump.—The Californian pump is essentially a chain pump. A rectangular box, about 10 in. $\times$ 3 in. inside measurement, and varying from 10 to 30 ft. in length, according to need, is traversed by an

FIG. 34.



CALIFORNIAN PUMP

endless flexible band or belt of canvas, on one side of which are securely fixed at intervals wooden discs nearly as large as the inside of the box. The lower end of the box is furnished with a roller, round which the belt passes, and is immersed in the water to be raised from the pit, while the upper end delivers the water into a trough or launder, by which it is carried away. At the upper end, the belt passes round a second roller or drum, which is made to revolve by either hand- or water-power. In Fig. 34 is shown one driven by a water-wheel: *a* is a flat wooden pipe or box, open at both ends, forming the pump; *b*, the pump-belt, carrying the wooden stops, faced with leather, called the buckets or suckers *d*; *c*, the ends of the belt, joined together by lacing; *h*, the drum fixed on the axle of the water-wheel *w* and turning with it; *i*, entrance of water pumped up; *e*, exit of same; *f*, launder or race to convey the water from the pump and wheel clear of the working; *g*, a sluice-box set in a head-race to bring the water necessary for driving the wheel.

Chinese pump.—A modification of this pump, known as the "Chinese" pump, differs from it only in being made entirely of wood, the belt even being composed of short pieces of wood hinged together by wooden pins.

Syphons.—Syphons may often be used for draining the workings

when it is not necessary to raise the water to a great height, and where the necessary fall for the delivery end can be had. Raymond describes one of unusual dimensions used in California. It was over 1000 ft. long and 4 in. in diameter. The pipe was made of No. 24 galvanised iron, in sections 30 in. long, riveted and soldered together. The water was raised 18 ft., and the outlet end had a fall of 40 ft., so that the delivery was 22 ft. lower than the inlet of the syphon. The two ends were fitted with 4-in. brass taps, which were closed when the syphon was to be filled. This operation was easily performed in about 2 hours by means of a 3-in. Douglas force-pump, throwing water in at the highest point through a vent-cock, by which also smaller quantities of water might be supplied from time to time to displace the air that gradually found its way in through leaks. An air-chamber at the bend was projected, but not found necessary, as by shutting the 4-in. traps at either end, it was easy to fill the syphons by means of the pump during the men's meal-hours. The flow was easily controlled by the taps, the lower or delivery-tap being usually left fully open, while the inlet tap was partly closed. The velocity of the current was such as to carry out tons of coarse sand and gravel, some of the stones being as large as walnuts, and the sluice-boxes set at the usual slope were kept half full of water. There was no trouble in keeping the water within 2 in. of the inlet or receiving end, and this was plunged to within 5 in. of the bottom of the shaft.

Modes of working alluvial deposits.—The apparatus employed in recovering the gold from the wash-dirt has been already described ; but it is also necessary to state that the manner of reaching the gold-bearing stratum or of raising the wash-dirt for treatment, is not in all cases alike, varying as may be dictated by the nature of the deposit. The essentially different methods are two—"Stripping," and working through "shafts" and "drives." Where the nature of the ground and the supply of water admit of it, hydraulicing is much cheaper than either ; but that branch of mining is so important as to require a special chapter.

Stripping.—This mode of working must be adopted when the alluvial matters overlying the wash-dirt are soft and friable, or saturated with water from underground drainage. It consists in stripping off and taking away the whole of the non-auriferous superstratum, often having a depth of 20 to 30 ft., and thus laying bare the pay-dirt. Operations are begun by bringing a tail-race or drainage-channel up to the workings at such a level as to draw off the water from the bottom. This done, a "paddock" or patch is opened, the superincumbent earth is removed and carted away, and the layer of wash-dirt is extracted and treated by sluicing, as already described, to separate the gold. The tail-race is carried on up the creek or washing, and carefully timbered, covered in, and protected, so that the waste earth may be thrown on it without fear of damage. Care must be

taken to prevent floods, by means of dams to confine the water to a regular channel. Sometimes pumps and syphons will be necessary to keep the workings dry. The most advantageous plan is to fill the first paddock or space with the over-burden or refuse earth from the second, and so on ; thus no part of it has to be removed any great distance, nor to be lifted to a considerable height. In one of the reports on gold-mining in New South Wales, a shallow placer is mentioned which employs 70 men and 45 horses, besides 3 powerful engines. One of these last is engaged in hauling up trucks of wash-dirt to the sluice-boxes, a second in hauling up trucks of refuse to the tips, and the third in pumping. The trucks are hauled up by a wire rope on a tramway

laid on an incline, and this application of the tramway has been found to secure an immense saving of labour.

A common feature in alluvial mining is that a shallow layer at the bottom contains all the gold, while a thick superposed mass carries none, or so little as not to be worth working. In such cases, stripping the top portion off is sometimes the cheapest way of going to work. To do this by hand labour entails heavy expense, and it will be often found more economical to employ machinery,



FIG. 35.

GRAB-DREDGER.

such as the modified form of grab dredger which Priestman Bros. make for the purpose, and which is shown in action in Fig. 35.

Working by shafts and drives.—In many respects this is precisely similar to ordinary mining for any mineral—such as coal, for instance—a shaft being sunk on to the layer of pay-dirt, and tunnels or adits driven from the bottom along the deposit in all directions. This can only be done where the earth possesses a certain degree of stability. Generally if not always, the shafts and adits require timbering, to keep loose earth and water from falling.

Slabbing a drift.—Difficulties may sometimes arise by the shaft tapping a drift or wet sandy stratum. The following plan was successfully adopted for dealing with a case of this kind on the Ballarat gold-field. The drift was cut away much larger than the proposed size of the shaft, slabs were then put in of the size of the shaft, and puddle-clay was thrown in and rammed down behind the slabs. In sinking deeper, however, a second drift was encountered, also containing much water, and this could not be overcome in the same manner as the first, on account of the impossibility of keeping the sand back while sinking through and

putting in the puddle-clay. The following plan therefore was devised. Four slabs were fastened together in a square of smaller dimensions than the shaft, placed in the centre of it, and driven down into the drift; the last set of slabs in the solid ground above the drift were then removed, the walls of the shaft were cut away a foot or more into the "country," and longer slabs were fitted in the place of those which had been taken out. The shaft was then cut down of the increased size and slabbed, the four small slabs in the centre being always kept below the lowest set of long slabs, and the drift between the box of short slabs and slabbed sides of the shaft only being removed so as to make room for the slabs until the set had been placed in position and made fast. The drift between the box in the centre and the long slabs was then taken out, the box was lowered, and the same process was followed with each set of slabs, until they got through the drift into solid ground again. They then commenced to slab upwards from the bottom with slabs the proper size of the shaft, and as each set was put in, puddle-clay was filled in behind, and rammed down, the long slabs, called "drift"-slabs, keeping back the drift. When completed, the shaft was the same size all the way down, and there was a wall of puddle-clay between the double sets of slabs, extending a foot or more above, and the same distance below the drift. This was generally effectual in stopping back the water, but in some cases the latter forced a passage through, and recourse was then had to caulking the seams between the slabs. These last were split, and at first were fitted into sets by notching the ends; but this plan was not suitable in wet ground, and gave way to that of boring holes through the ends of the side-slabs with an auger, and driving pegs in to keep the end-slabs in position. Instead of caulking the leaks, plugging was sometimes resorted to. This was managed by boring holes through the slabs with an auger, and then forcing through plugs of puddle-clay, prepared and rolled to suit the size of the holes, by means of a rammer, until it was impossible to force any more, or the leak was stopped.

Piling.—Another plan sometimes followed is that known as "piling." The shaft is "opened out," as already described, a little above the drift, a frame is made having nearly the same dimensions as the shaft where opened out, or enlarged and placed in the latter, and slabs pointed at one end are then driven down behind this frame as far as possible into the drift immediately below. Sinking is then continued in the drift, always keeping the points of the slabs driven into the drift below the spot where sinking is going on. When deep enough, another frame is put in to support the lower ends of the slabs against the inward pressure of the drift, and the next set of slabs are then driven down between the second frame and the lower ends of the first set of slabs, and so on throughout till the solid ground is reached again. The slabs thus driven are called

"piles." Sometimes, when sinking through such a drift, a box is driven down in the centre of the shaft, which helps to support the points of the slabs against being forced into the shaft. The greatest difficulty is occasionally experienced in keeping the frames in position, because the pressure from the drift may not always be equal on all sides, and consequently the frames and piles may be carried out of plumb; and now and then the drift will burst into the shaft from one side or the other, in spite of all precautions. As fast as this is removed, to enable the men to make good the timbering, more may run in, so that a large cavity is made on one side, and, the pressure against the frames, being unequal, may even twist them half-way round. Despite the twist, it is possible in some cases to continue sinking to the firm ground by filling up the cavity with bark, clay, and other materials, then, having drained the water from the drift to some extent, but before puddling up so far, the shaft may be repaired so that the slabbing and puddling up may proceed in such a manner that the shaft may be plumb when finished. In other cases it may not be possible to stop the running-in of the drift, nor to repair the damage done to the timbering; then the shaft may be filled up with earth to the level of the top of the drift, and a fresh sinking commenced from that point, taking out the old timber where necessary as the descent progresses. When the solid ground is reached, the slabbing and puddling up may be carried on as already indicated.

Drawing slabs.—For the purpose of taking out the slabs from a shaft which it is no longer intended to use, a stage is made to suit the shape of the shaft, and small enough to be raised or lowered inside the slabbing of the shaft. This is lowered into the shaft by ropes, and secured. A man is then let down the shaft by a windlass, and as fast as he draws out the slabs he sends them up by it. As soon as a few sets have been drawn out, the stage must be raised level with or above the top of the space from which the slabs have been drawn, so that in case of the sides thus left falling in, the earth might descend into the shaft below the stage.

Iron cylinders in lieu of slabs.—In consequence of the difficulty met with in sinking a shaft by the ordinary means through an unusually troublesome drift, the idea was conceived of using sheet iron instead of wooden slabs. It was formed into large cylinders the size of the shaft, the latter was cut down to the proper size, and the cylinders were let down one above another, with the intention, when the drift was reached, of forcing the cylinders into the drift in advance of the sinking; and it was reckoned that, by keeping the cylinders well driven down below the point at which sinking was proceeding, the drift would be prevented from running into the shaft. The cylinders answered very well in sinking through the solid ground so long as they had sufficient play;

but when the clay began to swell, they became fixed and could not be forced down. Cylinders were then prepared of less diameter, and let down inside of those which had become fixed, and thus a depth of about 30 ft. was reached; but where great pressure came against the sheet iron it buckled, and wrought iron would not stand the pressure necessary to force it through a drift.

Boxing.—Where the material overlying the wash-dirt is of a loose nature, it will be necessary to slab the drives or adits much in the same manner as the shafts. This operation is termed “boxing” in Victoria. “Sets of timber,” or legs and caps fitted together by means of grooves in the cap-piece and shoulders on the legs, are commonly used.

Paddocks and paddocking.—It has already been explained that the term “paddock” is applied to a patch of wash-dirt lying *in situ*. It has also another meaning. Sometimes, instead of washing the pay-dirt from a digging as fast as it is raised, the miners stack it in “paddocks,” which are spaces alongside the shaft, strongly fenced in with slabs, until the claim or property is exhausted, and then hold a general washing up. This system of working is called “paddocking.”

Working Reef-washes.—In the case of reef-washes or benches, the alluvial wash-dirt may lie 300 or 400 ft. above the present level of the stream, to which it must be conveyed for treatment. A simple method of transporting it is described by Ben. B. Spargo, of Snowy Creek, Victoria, and consists in shooting it in green hide bags down sawn timber shoots.

Sluicing.—The advantages of sluicing an alluvial deposit are manifestly great, as when once the races or water-channels are made, very little labour is required for carrying on the work. Under varying conditions, there will be varying results. In one instance, 6 men have been able to sluice away an acre a fortnight; in another, 8 men got through an acre a month. The following principles are recommended to be observed when it is intended to sluice away the whole length, breadth, and depth of an auriferous deposit:—

1. To get a proper and sufficient fall in the tail-races (or channel for carrying off the light particles of waste materials) for boxes of large dimensions.
2. To have a full flow of water to keep the tail-race clear and in working order.
3. To have small wash which will easily run off, and to strip the ground by water-power. Wherever this can be accomplished, a very small amount of gold will leave a profit.

Water-soaked flats can only be worked by a tail-race with sufficient fall and large boxes thoroughly draining the ground. Sometimes these tail-races have a very low grade or inclination. An important one

alluded to in the reports on the New Zealand gold-fields was carried for 8 miles at a grade of 1 in 62·11, and was then continued at 1 in 59·2, probably because the ground was favourable to the more rapid fall, which of course facilitated the removal of the sludge.

Covered tail-races.—For the following description and illustrations of the method of working with covered tail-races, first practised by the Chinese at Kyeburn, and since adopted by some Europeans, indebtedness is expressed to H. W. Robinson, Warden of the Mount Ida division of the Otago gold-field, New Zealand.

The Kyeburn river is a mountain stream, ordinarily of no great volume, that finds its way through a shingly bed filling the channel. It is subject to heavy floods, which come on with great suddenness but subside as quickly. The ordinary state of the river is that of a smallish stream of water meandering at a rapid pace through a wide expanse of shingly gravel. At a depth of a few feet below the surface of this gravel bed is a rather more compact layer of similar gravel, in which gold is thinly distributed. The problem was how to work this auriferous layer to advantage. When an open pit, called by miners a "paddock," was sunk in the shingle below the level of the flowing stream, the water filtering through the loose gravel quickly filled it up. It was possible to keep the water under by baling or pumping, but both these modes were too expensive. Tail-races, that is to say, open ditches carried through the shingle down the river, would drain the excavations, but these were liable to be lost at any moment, as whenever a flood came down they would be filled up and obliterated. To remedy these difficulties, the plan was devised of bringing up a tail-race to drain a paddock, to build it carefully as a covered drain, and to fill in the loose shingle over it at once, thus securing it from damage by floods. The method will be better understood by reference to Fig. 36.

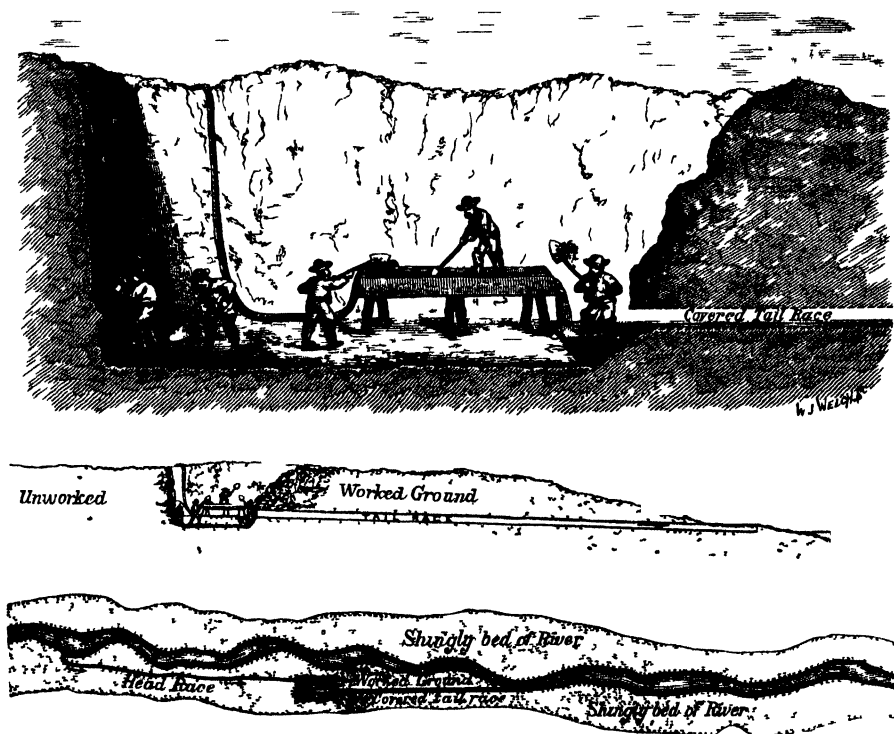
The gravel is thrown into a sluice-box, to one end of which water is conducted by hose or otherwise. The force of the water carries the gravel along. A man on the box keeps the gravel stirred with a sluice-fork, and throws out all stones large enough to be lifted by the fork. At the end is a slight drop, where another man with a shovel is constantly employed lifting the finer gravel, and only what the water can hold in solution or suspension escapes down the tail-race. The tail-race is lengthened as the workings proceed up the course of the river, all débris being heaped over it. The paddock is constantly moving up. All gold is saved in the sluice-box only. The tail-race is simply a drain.

The following description of a method of working low beds of gravel appeared in the 'Mining and Scientific Press' of San Francisco. A few miles from Ione, Amador County, is the Arroyo Seco mine, which, as the Spanish name indicates, is situated in the bed of a "dry creek."

The ground was known to be rich, but there was no fall, and no way of getting rid of tailings. The pay dirt is 5–10 ft deep, and there is considerable stripping to do.

In order to get rid of the worthless top dirt, a large crane or steam derrick was introduced. The two engines to operate this are 9 by 16 in., and there are two 48-inch upright boilers. They hoist 5 to 7 tons at a load, on the end of a boom 110 ft long. The dirt is hoisted 50 ft. high, swung to one side and dumped, and the tub is returned in $1\frac{1}{2}$ minutes.

FIG 36.



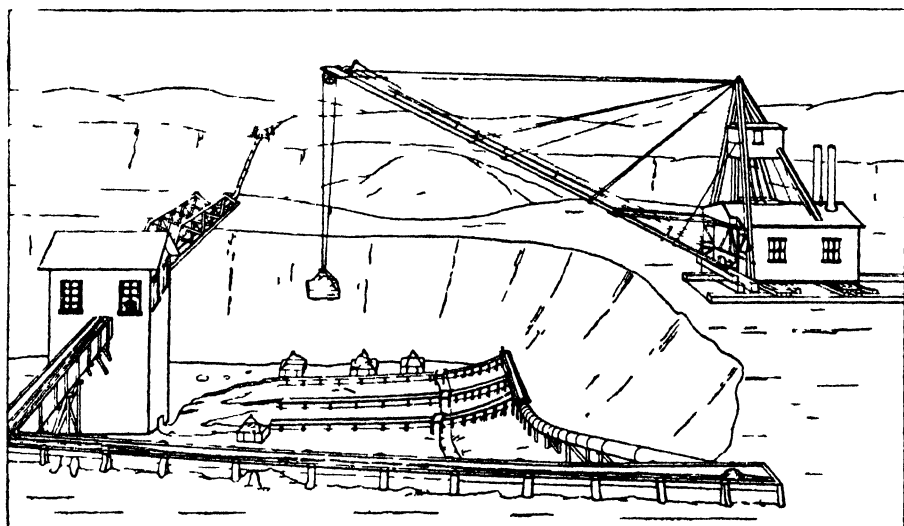
SLUICING THROUGH COVERED TAIL RACES

The derrick house, containing the engines and boilers, is on tracks laid on the ground, so that it can be gradually moved up the creek as the claim is worked. With this derrick they can dump the "strippings" 110 ft on each side of the centre, allowing the men to get the pay gravel. In addition to the engineer, one man controls the movement of the bucket and derrick, and one man on the boom attends to the dumping.

The pay dirt is washed in sluices, as shown in Fig. 37. The tailings, small rocks, sand, and water run into a general sump under the tall house. These tailings and the water have to be removed, as there is no fall and no way to get rid of them except to hoist them out of the way. This is accomplished in a very simple but ingenious manner.

In the sump are two submerged centrifugal pumps, of peculiar pattern, and designed for this particular work. Each has two 11-inch discharge pipes, the capacity is 600 miners' inches of water, or 900 cub. ft. per minute. These pumps have no steps or bearings made under water, the whole weight resting on two 26 inch anti-friction wheels.

FIG 37.



MINING LOW GROUND WITHOUT FALL.

On the floor above these centrifugal wheels are two 15-inch Knight water wheels fed from the main supply pipe with 68 ft. head. These wheels run the submerged centrifugal pumps below and raise the sand, gravel, and water from the sump, throwing the whole *débris* and water into the flume seen issuing from the side of the pump-house. From this flume the gravel and rocks are shovelled out by men, the water is screened and again returned by a pipe, to be used in sluicing; 68 ft. of water does the washing, pumping, and all.

The centrifugal double discharge pumps will pump sand, gravel, chips, leaves, or anything that will pass through a 3-inch mesh. They have pumped up with the water sand and fine gravel as fast as two men could shovel it to the pump for hours at a time, and raised it 36 ft. high.

These pumps have been in constant use for several years, and it can not be seen that they are any the worse for the hard usage.

Dry washing.—This somewhat contradictory term is applied to processes for separating the gold from the dirt without the aid of water. One is practised, principally by Mexicans and Indians, in the Western States of America, where a supply of the liquid element is unobtainable. Only the richest pay-dirt is treated by this process, which is conducted as follows. The dirt is spread on a raw hide, and allowed to become perfectly dry; next it is powdered up by rubbing it with the hands, and all the larger pebbles and bits of stone are picked out and cast away. The residue is then placed in a *batea*, already described, and the lighter particles of base matters are winnowed from the gold by tossing the contents repeatedly into the air with a circular motion, which enables the wind to disperse the light bodies while the heavier gold falls back into the *batea*. When this has been continued till the heavy residuum consists largely of the precious metal, the purification is completed by blowing, which process has been alluded to on a previous page. When two men or more are working in company, a hide or blanket may be used for the winnowing. Nothing but very rich stuff can be worked remuneratively by this system, and the miner never goes very much deeper than his own height in search of the pay-dirt. Frequently, instead of stripping off the worthless overburden or overlying stratum of earth, he sinks a little shaft or pit about 6 to 8 ft. deep, and then drives or burrows on the wash-dirt beneath. This system of working is known in America as "coyoting," from the resemblance of the diggings to the underground houses of the coyote, and is not always confined to the operations of the dry-washing miner. Where water can be had at some seasons but is absent in others, the diggings are often called "gulch claims." Such placers are now in California usually worked only by Chinese.

A form of winnowing-machine for separating gold from sand, called the "Eureka concentrator," has been introduced for working dry placers, which cover a boundless area in Tropical America, Australia, Siberia, &c. It weighs about 250 lb., and measures 3 ft. by 4 ft. by 4 ft. The ore-table is stationary, not having a side or end shake of any kind. Under it, is a peculiar double-acting bellows, which forces a pulsating blast of air up through the canvas, which, on top of a wire frame, forms the ore-table. Riffles are formed on this inclined table to catch the heavier particles. A peculiar and valuable feature of the appliance is an apparatus which keeps the coarser gravel, &c., moving on downward to the discharge end. The machine is constructed to stand the dry hot winds of Arizona and New Mexico, and has no leather or boards used in its construction. By simple means, feed- and tailing-elevators are

attached, by which the hopper is fed with the sand or gravel, and the tailings are removed out of the way. With these attached, a mule or horse runs the machine, doing the work of 3 men. The No. 1 machine is equal to 40 in. of water in sluices, and works dirt as cheaply as sluicing with water at $2\frac{1}{2}$ c. ($1\frac{1}{4}$ d.) an inch. Light machines are made for a man to run by hand. The apparatus works very rapidly, and has a capacity of 3 tons per hour. Ordinary placer-gold is readily caught in its riffles. The concentrator is quite simple in construction, not liable to get out of order, and well adapted for its work. More or less blast can be given to work heavier or lighter stuffs. A hard graphite lubricator is used on the journals, and is applied in a peculiar way. This is of great advantage in hot climates, where ordinary lubricators do not answer. There are in these countries thousands of acres of ground which cannot be worked except with machines of this kind, but which can be made available when worked dry.

The *Colonies and India*, of June 2, 1882, records that the Mines Department of New South Wales "has tried a new dry-blowing machine imported from California. About 4 dwt. of gold was mixed with 50 lb. of earth; in 3 minutes the dirt was fanned off, and it was then found that less than 1 gr. of gold was lost. This machine will be very valuable in places where water is scarce."

Other forms of apparatus have been introduced, in which mercury plays an important part. The one which has been most prominently before the public operates by throwing the auriferous material by centrifugal force against a wall of mercury (why not amalgamated plates?). The material is placed in a sort of hopper, from whose taper end it is allowed to pour between two curved copper plates, situated horizontally and revolved at a speed of 65 to 80 revolutions a minute. Passing between these plates, the sand is driven by centrifugal force against a "wall" of mercury, which is also maintained through the action of centrifugal force. Surrounding the copper plates is an iron casting like an enormous stew-pan, with the greater part of the bottom cut out, so that its half section resembles a letter L. This is connected with an axis by 4 hollow arms, each of which is made to contain about 50 lb. of mercury. The whole arrangement is revolved in a direction contrary to that of the plates, and, by the rotary force, the mercury runs from the arms through grooves provided, and climbs along the upright surface, corresponding with the upright stroke of the L, thus forming a perfect wall. Every particle of gold as it is thrown against this wall instantly amalgamates with it, while the sand is partly thrown off by its own rebound and partly blown away by a series of pipes attached to a blower used in connection with the machine. When whatever quantity of sand the operator chooses is passed through, the machine is stopped,

the mercury is withdrawn by a tap from the arms, into which it falls the instant the rotation ceases. The operation results in the recovery of the gold without sensible loss, and the advantages claimed for the plan are :— (1) that it extracts every particle of gold from the matter passed through it, and (2) that it does so without serious loss of mercury. It is claimed that it will render it possible to work over again with profit the “tailings” of mines operated under old systems.

There is no available information as to the working cost of any of these methods. Their success depends entirely upon their ability to perform the necessary operation on a large scale at a cost not exceeding $2\frac{1}{2}d.$ to $5d.$ a ton. When there is any tendency in the material to cake together, either from presence of alkali or other causes, dry washing is out of the question.

River-mining.—Before detailing the methods adopted by modern miners, it will be interesting to glance at the contrivances which have been and still are relied upon among the indigenous dwellers on some auriferous river-banks.

In Ladak, a remarkable practice consists in pegging down the skins of animals with the hair on them in the beds of gold-bearing streams. The hair, presumably set against the current, acts the part of the modern miner's blanket, arresting the small particles of metal, which can be collected by drying and shaking the skins. This same practice is said to have been in vogue anciently in Saxony.

A modification of it is employed in Upper Burma, where horns of the wild cow (*Bos sondaicus*), with the hair on, are planted in the beds of rivers to arrest the gold, and, when charged with spangles, are sold for 12 or 13 *ticals* (1*l.* 12*s.* 6*d.* to 1*l.* 15*s.*) each.

Gold-washing in the Ribeirão de Ouro Preto, Brazil, is thus performed. The gold-washers (*faiscadores*) are dressed in a leather jacket, and have a round bowl cut out of the wood of the fig-tree (*gamelleira*), from $1\frac{1}{2}$ to 2 ft. in diameter, and 1 ft. deep (*gamella*, *panella*, *batea*), and a leather bag fixed before them. They generally select those places in which the river is not rapid, where it makes a bend, and has deep holes. They first remove the large stones, and upper layers of sand, with their feet or their *gamella*, and then take up a bowlful of the deeper and older gravel of the river (*cascalho virgim*). They continue to stake, wash, and strike off the stones and sand at the top, till the heavy gold-dust appears, pure, at the bottom of the vessel, on which a little water is thrown in with the hand, and the gold is at length put into the leather bag. This mode of gold-washing is here called *mergulhar* (“diving”). They sometimes wash the *cascalho* upon a platform (*canoa*), erected on the spot.

The method of collecting the auriferous sand and detritus from the

beds of the deep pools in the rivers and streams of the United States of Colombia is by diving, and scraping it together with the *almocáfre*, and thus loading the *batea*, which they bring up from the bottom, and, after carrying it to a convenient place, wash the contents. The contrivance of the native for keeping himself at the bottom of the pool while engaged in collecting the detritus and sand from among the boulders and out of the crevices in the formation, is very rude, but effective. It is as follows :—He selects an oblong quadrangular stone of the requisite weight, say about 10 to 12 lb. One side of this he makes concave, to be adapted to the nape of the neck, and then about it he works a netting of *bejuco*, which is the natural rope of this country. To this he attaches a broad flexible band of tough bark by both of its ends, and of such a length as to be slipped readily over the head from behind forwards as far as the forehead, while the stone is resting upon the nape of the neck. With the stone thus arranged upon his neck, and armed with the *almocáfre* and *batea*, he cautiously wades to that depth where he can retain his foothold with his head above water ; then, throwing himself forward into the water, the weight of the stone bears his head downward to the bottom of the pool. Here he secures the *batea* stationary by putting a stone into it, and then groping about, with *almocáfre* in hand, he scrapes together the auriferous sand and fills the *batea*. Before his *batea* is loaded, the diver is compelled, perhaps several times, to rise to the surface for fresh air ; and this he does by slipping the band from his forehead and casting off the stone. After rising to the surface, he swims to the place where he can gain his foothold as before, and then drawing towards him the stone, by means of a cord attached to it, he replaces it upon his neck, and after he is sufficiently refreshed, dives in the same manner again. The *batea* is necessarily always retained where the depth of the water is not greater than the diver's height, as it would otherwise be impossible for him to raise the *batea* full of sand and detritus. By this rude and tedious management does the gold-washer obtain the gold from the bottom of the pools, which have not, generally, a greater depth than his own height. In some instances, however, the beds of pools of considerable depth have been partly washed by some natives more daring and persevering than the rest, where the great abundance of gold has been an inducement. In these cases, the diver only attempts to collect some lumps of gold from the crevices in the strata, and from among the loose rocks, or to gather a small quantity of auriferous sand in a *tortuma* or gourd, which is repeatedly emptied into the *batea* on the bank, and, when a sufficient quantity is collected is there washed.

Two distinct kinds of river-mining are in vogue among colonial gold-miners one being prosecuted by emptying the river-bed of its water

before commencing operations, the other conducted while the river is in the occupation of its bed. The former consists in turning the stream out of its course by means of a dam into a new channel, which may be either a ditch cut in the ground, or a wooden flume. Of course, only small brooks and rivulets can be treated in this way, and the working even in them can only be carried on with safety during the dry season, when the water is low and not liable to sudden flooding, as a freshet might carry away tools and implements, and undo all that the miner had done, in a very short time. Moreover the little tributary streams from the hills, which come in on every hand, give a deal of trouble, and necessitate pumping. Arrangements must also be provided for removing the huge boulders which often encumber the bed of a stream, and hide much of the gold. The pay-dirt is excavated and run through a sluice to extract the gold. The drawbacks and risks connected with this kind of mining have to be taken into consideration, but it has sometimes yielded very remunerative results. In California, the rich river-bars have been all but exhausted; yet as late as 1875, some Chinese were washing the banks and bars of the Yuba river in this way, and making it pay, though the same ground had already been worked three or four times by whites. Fig. 38 illustrates the river claim of Trent, Diamante & Co., which is worked in this manner. Guinness speaks of the plan as being most successfully used when reaches of some extent and of small depth occur, and when only a moderate depth of shingle overlies the solid rock forming the bed of the river. Considerable areas may sometimes be made available by cutting a new channel through an isthmus, and working the bed of the bend thus laid bare.

Lifting rivers.—A plan known as “river lifting” is practised in British Columbia, and elsewhere, also termed “fluming,” the river being removed from its bed by means of a flume. It is applicable to streams not exceeding 10 to 15 ft. wide and 2 ft. deep, and consists in providing a temporary artificial bed for the stream to flow over while the actual bed is being worked out. Two essential conditions are,—an abundant and cheap supply of timber, and an adequate fall in the stream. The first step is to drive stout piles into the ground on both sides and in the centre of the stream at a few feet apart, joining the members of each side row by means of stout quartering spiked to their upper part. Joists are then laid across the river with their ends resting on the lateral bars, so that each triplet of piles is joined transversely. Next, planking is nailed on the transverse bars inside the piles and carried up to the tops of the piles themselves, forming a wooden sluice or launder just above (say 2 or 3 in.) the stream, and of a capacity to accommodate the stream. When the sluice, which may be $\frac{1}{2}$ or $\frac{3}{4}$ mile long, is complete, the river is lifted into it in the following way. At the upper or inlet end of the

sluice, the stream is gradually dammed by driving piles in so that their heads are flush with the floor of the sluice. The piles should be commenced with spaces between, and these gradually filled up, so as not to concentrate the stream into one narrow channel. As soon as the damming is completed, the stream, unable to flow in its old bed, gradually rises till it enters the sluice, and then, if the fall is sufficient, it fills the sluice and flows bodily through it. While the river occupies the sluice, the bed is worked out and cradled or sluiced. In case of a

FIG 38



DRAINING A RIVER CLAIM BY CALIFORNIAN PUMP

freshet occurring, the damming piles may be pulled up and the stream allowed to re-occupy its bed, thus reducing the risk of having the sluice destroyed. This does not prevent the dam being reconstructed and work resumed when the freshet is over.

The banks of the creek or stream must be sufficiently steep to prevent the water running away in a lateral direction; when this is not the case, the stream has to be banked in for some distance above the dam.

Egleston describes a plan adopted on Feather River, in Butte County,

California, on the Cape claims. "Two wing dams were constructed across the river; all the water above the first dam was carried by a large flume to the lower one, and the water between the two dams was pumped out by wheels run by the current in the flume. The gravel on the river bed was washed in sluices, and during 42 days yielded \$680,000. Only one-third of the ground had then been washed out when the flume was destroyed. Such methods can only be employed on small streams in the dry season when there is the least water to contend with. The stream must be diverted from its course or pumped out. Such methods, as shown by the example cited above, are full of risk. They seem very attractive when the rich pay-dirt is brought up from the bottom and the calculation is made of how much gold is contained in the river bed, but it is seldom worth while to flume them for a short distance,—generally it will not pay to do it for less than 300 to 400 ft., and often more must be flumed,—and when all the water has been pumped out, derricks run by hurdy-gurdy wheels or by hand power must be erected to remove the large rocks, to get at the gold beneath them, with no certainty that the whole of the work will not be swept away by a sudden rise in the river."

Shaft and Drifting.—"When such a method is not practicable, the river may be explored to find the bars and rich placers. When they are located, a shaft is sunk to the depth of 40 to 60 ft. above high water-level, and a drift is run in the rock under the bars. The gravel is taken out from above, hoisted to the surface, and washed in sluices. The water pumped from the shaft, can, if necessary, be stored for washing. Gravel containing only 70c. (35*d.*) a cub. yd. is worked in this way." (Egleston.)

Booming.—"Where the quantity of water is, or is likely to be, small, in order to economise it, it is the custom to dam the stream, and, when the water has accumulated sufficiently, to open the gates and allow it to rush out against the gravel with great force. This process, called "booming," is sometimes carried on by hand power and sometimes automatically. When the process of "booming" is carried on automatically, it is effected with the use of the so-called "shot-gun dam." This is an automatic discharge gate, so constructed as to have the gate of the dam lifted at intervals by the water itself; to do this the gate is attached to one end of a suspended beam, which has a water-tight box with a valve in the bottom, attached to the other. When the dam is full, a part of the water flowing over passes into a launder, arranged for the purpose, which conducts the water to the box. This is so adjusted that when it is full it lifts the gate by itself sinking. The contents of the reservoir is thus instantly released, and flows with great force and rapidity to the point where it is to be used. After it has fallen a certain distance, the valve of the water box comes in contact with an upright post, which opens it and lets the water out. The empty box rises and closes the gate. This is

repeated at regular intervals, giving a heavy head of water and a rapid current every few minutes. This method has been extensively used both in Colorado and California. Before this method was invented, the gate was always lifted by hand and required the constant attendance and care of one man. By the present method, the gate requires to be looked after only occasionally. The work is entirely automatic, the number of times per hour that the gate is lifted being perfectly regular, and depending only on the supply of the water." (Egleston.)

Pumping.—"It has been proposed, and a great many attempts have been made, to raise the gold contained in the river gravels from the bottom of the streams with powerful pumping engines, capable of raising 25,000 cub. yd. in the course of 24 hours, half of the bulk being water. It is claimed that these pumps are cheaper and better than any dredging machines. Notwithstanding these assertions, however, neither dredging nor pumping has proved successful, for the reason that the river can only separate the gold in the same way that a sluice does. The idea of most persons not familiar with river sands, is that the sand is deposited in regular stratified layers, and that the gold is disseminated regularly through them. No one who has any experience in sluice mining would believe this, for the whole object of the sluice is to provide places where the gold may be caught in mercury, the bottom of the sluice being arranged with reference to this. At the bottom of the river the sands are always unequally distributed. The bed-rock is never even, but is always filled with crevices, into which the gold falls by gravity and is not accessible to the dredge. Where the river flows over a soft gravel, and the bed rock is either turned on its edge or is soft from decay, it is well known to gravel miners that the gold will penetrate the bed rock sometimes for a considerable distance, so that it is frequently worth while to take it up for some depth. Dredging and pumping machinery must do their work progressively and systematically, starting from a fixed point and proceeding toward some other. It is impossible for the machine to find where the gold lies, and even if every spot were known beforehand, the irregular nature of the deposit, and the fact that the bed-rock is often the richest part of it, would make it impossible to work a spot here and there, so that in pumping or dredging, the barren material will be more likely to be raised, even supposing the gold is actually there, than that which contains it, as the gold is hidden away in pockets beneath that ground which can be reached by the dredge. These processes, therefore, have not been successful, and it may be stated as a general proposition that if the river bed is poor it will not pay to work it by any method, and if it is rich the usual process of damming and fluming, which has sometimes proved successful, with all its uncertainties, is probably the best, as it is certainly the cheapest method of working it in dry seasons when the

flumes are properly constructed. In such a case, the practised eye of the miner distinguishes the points where the gold is likely to be found, and works those points only, leaving the barren material behind ; but the machine must take poor and rich as it comes, and cannot in any case work the bed-rock." (Egleston.)

Atmospheric Cylinders.—Another plan of removing the water from the bed to be worked is by means of atmospheric cylinders. This system is described by Warden Allen, of Picton, as follows. "The cylinders are large enough for a man to work in. They are made in lengths, the length used depending upon the depth of the river. A powerful air-pump is attached to the apparatus, and air is forced into the cylinders when they are placed in the river. The air forces out the water, and the man in the cylinder is able to collect the drift for washing. I am told it is very trying work for the men. Dredges would answer better."

Subsequently, Warden Allen kindly furnished the following additional particulars. "I am inclined to think that the apparatus is useless for all practical mining purposes, except probably prospecting shingle beds of shallow rivers. I am informed as follows :

"That the apparatus consists of 2 powerful air-pumps, a sufficient quantity of indiarubber tubing, and iron cylinder in lengths of about 3 or 4 ft. each, and a crane and windlass for raising and lowering the cylinder into the river. The cylinder is composed at the top of an air-tight compartment about 5 or 6 ft. high, of the same diameter as the cylinder (about 3 ft.) ; to this upper compartment the tubes connected with the air-pumps are attached, and there is an escape-valve to allow the air to escape when the pressure is found too great. On the floor of this upper compartment is a man-hole that can be opened and closed at any time, and of course must be so fitted as to be water-tight. To this upper compartment, the workmen can affix by rivets as many lengths of iron cylinder as they require, according to the depth of the river. These lengths of cylinder are simply plain iron tubes, about 4 or 5 ft. long and 3 ft. in diameter. When a sufficient length of cylinder has been attached to the upper compartment, the apparatus is lowered into the river and rests upon the bottom. The men who are to work enter the upper compartment through a man-hole in the side or top, and the lid is replaced and fastened by screw-bolts so as to make the compartment water-tight. The inner man-hole in the floor is then opened, and air is pumped into the cylinder in sufficient quantity to force out the water in the cylinder, and leave the bottom of the cylinder as nearly as possible free from water. One man descends, and a bucket fixed to a block and tackle inside the upper compartment is also lowered. The man below fills this bucket with the wash-dirt (gravel or mud), and it is hoisted up to the

upper compartment. When sufficient dirt has been raised, the apparatus is drawn up, and the proceeds are landed to be cradled or cleaned up in any way adopted by the parties at work. The escape valve in the upper compartment previously mentioned is used as a regulator to protect the men at work, for too great a pressure of air would cause serious consequences to them.

"As I before informed you, the apparatus has not been at work for many months, in fact has not been properly tried, and is not likely to be used now, for a new company has been formed to divert the course of the river, and work the ground in the ordinary manner, and this apparatus is to be set aside. I am inclined to think it would not pay any company to work this atmospheric cylinder."

Dredging.—Rivers proper cannot, of course, be turned in this way, as the expense and risk would be greatly magnified; they are, however, worked by dredging up the bed without disturbing the course of the water. Dredgers built much after the fashion of harbour-dredgers and driven by a current-wheel, have been used for a long time and with considerable success, especially in New Zealand, where one at work on the Clutha river yielded 4*l.* a week each to 4 men for over 5 years; but they are liable in a certain degree to be interfered with by floods, they cannot work in eddies (where the gold is largely deposited), and their effective capacity is limited.

Warden Allen, of Picton, to whom the author is indebted for many kindnesses, sends the following account. "I have obtained from Mr. Warden Carew, of Lawrence, the following description of the dredgers used in the Molyneux and Clutha district.

"There are three water-power wheel-dredgers on the river, answering admirably, and giving very good returns. So far as I can learn, these dredgers are on the ladder and bucket principle; 25 buckets and sometimes more, according to the length of the ladder used, are mechanically connected by a heavy link chain revolving round a series of rollers attached to the ladder, acting as auxiliaries to two large tumblers, the heaviest of which, weighing 6 cwt., is attached to the bottom end of the ladder, round which the buckets revolve, and carry up the auriferous gravel. The tumbler at the top of the ladder discharges the gravel into a shoot emptying into a cradle rocked by hand. This primitive method for saving the gold could in my opinion be easily improved upon by means of excentric wheels to suit the different strengths of the current putting a series of shaking tables in motion, say, one with a contra action to the other. If this system of saving gold could be adopted, and I can see no reason why it should not, it would in my opinion be a much more economical method, dispose of more gravel, and save a great deal more gold.

"These dredgers can raise 100 to 160 tons of gravel in 24 hours, and the depth of water in which they can work is 4 to 12 ft.; length of ladder, about 37 ft. The greatest depth in which they can work in ordinary currents is about 20 ft.

"The average strength of the current is about 6 knots an hour by the log, but the dredgers can work with greater safety and more regularity in a 3-knot current.

"I believe these dredgers are the most suitable for this river, as the gold principally is obtained in the current, and there is little or no gold where the river is sluggish; 5 gr. of gold per ton will pay good wages, and there are miles of such ground in the river. Although there are some rich patches, there is no regular or distinct 'run' of gold. Generally the gold is sparsely disseminated through the gravel. (The foregoing description was obtained by Mr. Warden Carew from Mr. Nicholson.)

"Mr. Carew adds as follows. In addition to the information he has given, I may add that a dredger is formed with two flat-bottomed boats, each of about 12 tons, and that the ladder upon which the buckets revolve is fixed between the two boats. The buckets are made of iron, steel-tipped, and lift 1 to $1\frac{1}{2}$ cwt. each. There are 4 current-wheel dredgers in the river, besides two steam-dredgers of 25 and 22 tons (ordinary small steam-boats specially fitted up for dredging), and there are two new dredgers being built, one of iron, and it is said will be fitted with a different kind of machinery, costing when complete about 6000/.

"The larger of the 2 steam-dredgers now working is 10 horse-power, lifts 50 tons of drift per hour, and puts it through the cradle. The cradle is 13 ft. 6 in. long by 4 ft. 6 in. wide, and in addition has wide sluice-boxes from tail of cradle to the water. This boat has not been long in operation, but the miners estimate that when fairly at work in a good position 1 gr. of gold per ton will pay.

"Dredgers working by steam-power possess manifest advantages in being independent of the current; but even with them, a great impediment has proved to be that the least flood in the river brings down an immense quantity of tailings and other débris, which fill up the dredging-buckets to the exclusion of the much more highly auriferous older deposits on the bottom."

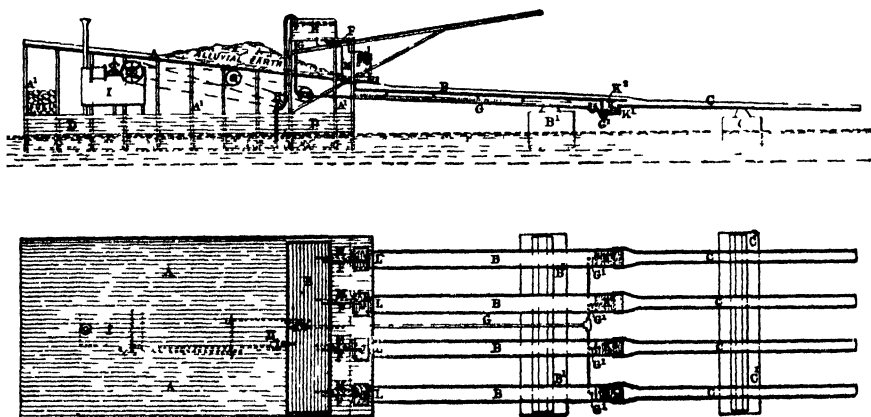
Fig. 39 illustrates a new sluicing apparatus, the invention of G. B. B. Elliott, of Melbourne, intended to be worked in connection with a dredger. Both Elliott's machine and the dredger are floated on punts, each 40 by 20 ft., and 6 ft. high, on the river intended to be worked. The dredger raises the wash-dirt at the rate of about 60 tons per hour, and deposits it on a sloping platform, whence it gravitates to a riddle upon which falls a constant shower of water, raised from the river by a small engine. The water washes the minerals, sand, and small gravel

through the riddle into sluice-boxes fitted with ripples, blankets, &c., in the ordinary way. The riddle is on hinges, and can be tipped up by means of a lever whenever it is desired to throw back into the river the large stones, &c. which cannot pass through. At the end of the first row of sluice boxes there is a box with a false bottom of finely perforated metal. A pipe, leading from an elevated tank, conveys water into this box. The pipe is fitted with a tap, which is turned off and on alternately by a pulley driven by a belt. This causes the water to flow spasmodically into the box and up through the perforated plate, imparting, at the same time, a lifting or pulsating motion to the wash. As the lightest portion of the wash comes to the surface first it is carried away, while the fine mineral particles, if any have escaped from the sluice boxes, fall to the bottom.

The cost in Melbourne, of the entire apparatus, including the dredger the sluicing machine, the two punts, engines, pumps, sluice boxes, and all other necessary appliances, is 2000*l*.

For gold mining, it takes 15 hands to work the apparatus. This includes two men for procuring firewood. The dredger itself can be worked by one man. All the men can live in the punts, which are constructed to float, if necessary, in 18 in. of water; but, if required, the dredge will bring up stuff from a depth of 45 ft.

FIG. 39



ELLIOTT'S FLOATING SLUICE.

References to Drawing.—*A*, inclined platform; *B*, sluice boxes; *C*, tail races; *A*¹, framing for the platform supported on punt *D*; *E*, elevated water tank; *F*, *G*, *M*, series of supply pipes therefrom; *H*, centrifugal pump; *I*, engine. The pipes *F* lead to perforated trays *F*¹; and the pipe *G* with its branches *G*¹ lead to the wells or troughs *K*, each of which has a perforated false bottom *K*¹; while the pipes *M* lead to the

heads of the respective sluices, over each of which is a perforated plate *L*. *B*¹ and *C*¹ are the buoys or punts for the sluice-box and tail race respectively. *K*² are hinged perforated plates for preventing large stones from falling into the wells or troughs *K*.

The mode of operation is as follows :—The alluvial earth to be treated is lifted and deposited on the platform *A* by dredgers. This earth is then conveyed to the perforated plates *L*, either by shovelling or by means of a water jet playing upon it, and as it falls on to the plates it is subjected to the action of a shower of water from the upper trays *J*, which washes all the finer particles through the perforations in the plates *L* into the sluice boxes *B*, which are of ordinary construction, the large stones remaining on the plates being then thrown out from time to time as required. The tailings from the sluice boxes are subjected to a water pulsation or jiggling in wells or troughs *K*, and the waste passes away through the tail races *C*, which are made sufficiently long to prevent the tailings from returning into the excavation being made by the dredger. When all the valuable alluvial earth has been excavated in one spot, the anchors or moorings are loosened, and the whole apparatus is floated farther up the stream and again moored, when operations are commenced afresh. By this means the operator is enabled to work river and creek beds, and to make artificial canals into alluvial flats on the banks of such rivers or creeks where otherwise it would be impracticable by reason of the difficulty of getting rid of the tailings at a sufficiently rapid rate.

A steam vacuum dredger was used at Oroville, on the Feather river of California, some years since. Skidmore saw the machine in operation at Oroville, Butte co., at that time, and thinks that if there had been any coarse gold or nuggets in the matter raised, it would have brought them to the sluice-box. He smelted the resulting gold, and found the nett proceeds to be about \$16 50c., after an expenditure of \$100,000. As a gold-mining appliance under the conditions of American streams (gorged with tailings to a depth of 20 or more ft.), it must and did prove a failure, for reasons too obvious to enumerate ; but Skidmore thinks that under conditions which may be imagined, but which are never found to exist, it would prove a success, viz. given a virgin river bottom, with 6 to 10 ft. of gold-bearing gravel on the bed-rock (averaging about \$3 per cub. yd.), and with water enough to float the scow and her machinery. There are no such streams in America, and it may be said that there at least it is a practical failure.

A form of dredger well adapted for dealing with gold and tin deposits in the beds of rivers, is that made on the grab principle by Priestman Brothers, of London and Hull. Dredgers of this class have been supplied to the French (Nechi) Gold Mining Co., of the United States of Columbia ; to the Long Reach Tin Mining Co., of Tasmania ; and for

the working of other alluvial gold and tin deposits. The illustration, Fig. 40 shows the dredger in actual work on the Nechi river. The grab having penetrated the material to be lifted, compresses it in the act of closing, thus securing any minerals contained therein, and which under other circumstances might be washed away. The dredger can be

FIG. 40.



PRIESTMAN'S GOLD AND TIN DREDGER.

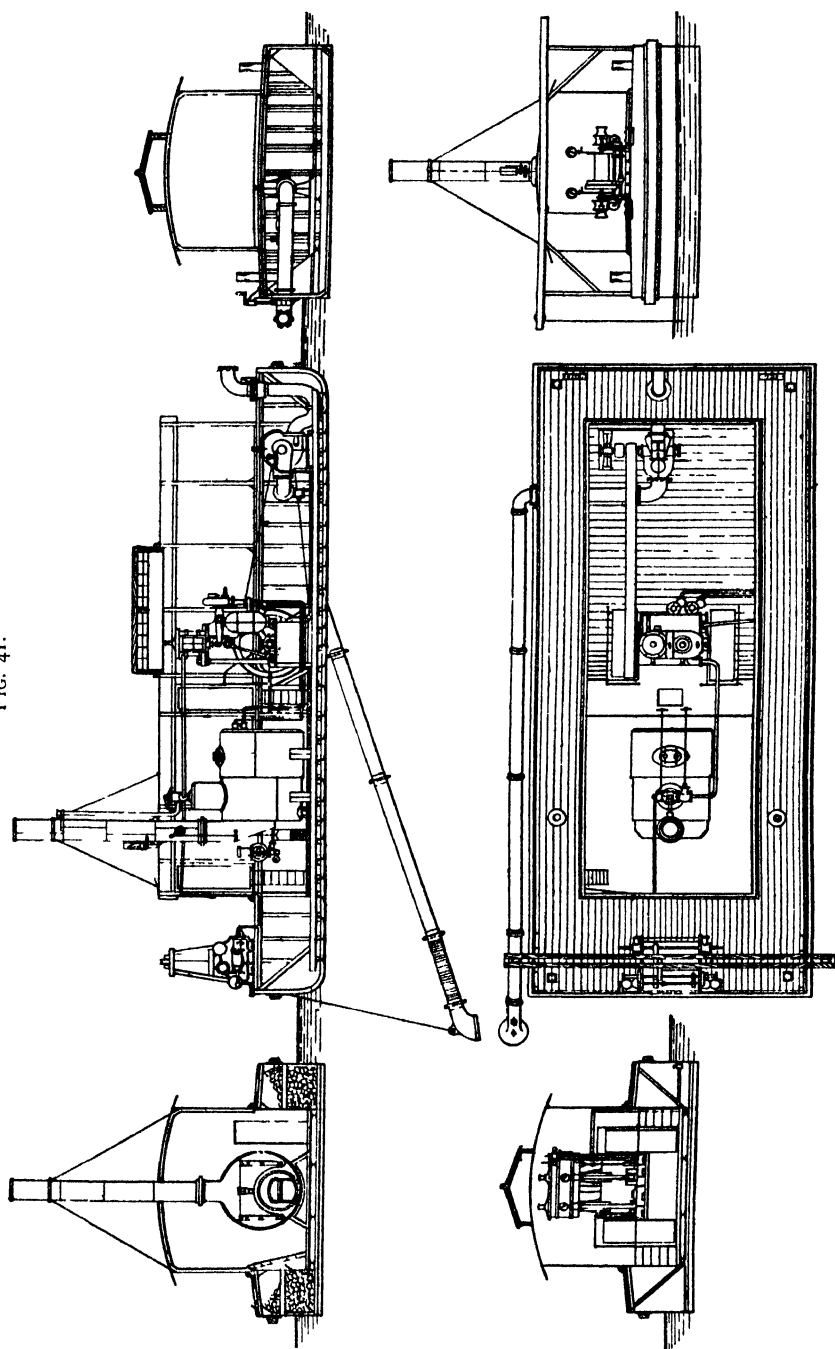
worked from a common barge or pontoon, and the fact of its being able to penetrate to any ordinary depth without stopping for adjustment is a great point in its favour. It will also easily lift pieces of tree trunks, boulders, &c., which are often met with in river beds, and would impede or even incapacitate an ordinary ladder bucket dredger. The simplicity of the grab dredger commends it to miners, as the wear and tear is small, and the dredger is very portable.

Fig. 41 shows various sectional views of a Ball dredger, similar to those which are being used on the Clutha or Molyneux river and other streams in New Zealand, on the Piranga river in Brazil, and in other places, for recovering the gold lying in the beds of running rivers, or on sea beaches, and in each instance with great success.

Though not connected with gold mining, it may be interesting to know that the biggest dredger that will be in existence is now being built on the Ball system, for the Durban Harbour Trust, Natal—it will dredge 1000 yards per hour, and has two 18 in. suction pipes.

The special advantages claimed for the "Ball dredger," as a means of working alluvial deposits of gold in rivers, lakes, swamps, and beaches, are the following:—The first cost is considerably less than that of a

FIG. 41.



BALL'S CENTRIFUGAL-PUMP DREDGER.

bucket dredger; wear and tear are also very small. It requires no special form of craft to carry the dredging machinery, which may be placed either on a raft, barge, or pontoon, as most convenient; or it may be erected on land, and the pipes led into the desired position at a considerable distance from the pump. Dredgers have been built drawing only 14 in. of water. Owing to the powerful suction action of the pump, the shape of the nozzle, and the flexible nature of the piping, the auriferous deposit can be brought up from holes and ledges in the rivers, &c., which a bucket dredger cannot touch. It can also work in greater depths than are possible with a bucket dredger, 100 ft. and upwards being quite within its powers when specially provided for. The material can be delivered to long distances (say 200–300 yd.) by the force of the pump, thus enabling the sluicing operations to be carried on where most convenient; and it can be raised to a considerable height above the level of the pump, say 10–15 ft. (or even more if extra power be applied), thereby providing a fall for the tailings. The material is well pounded and broken during its passage through the pump, and is delivered mixed with water in such proportions as the operator may desire—or water only can be discharged, by merely raising the nozzle off the bottom. A very small number of hands is required to work the plant, which is consequently capable of performing the maximum amount of work at a minimum of cost.

The essential feature of the Ball dredger is the use of a patented centrifugal pump, with a self-adjusting fan, partly made of metal and partly of rubber, which accommodates itself to the presence of any amount of grit, gravel, or stones in the material to be dredged, without showing any appreciable amount of wear.

The pump is so constructed as to pass stones of any size that will find their way through the pipes, though in practice a grating is generally fixed to the nozzle.

The connection between the pump and the ground to be dredged is made by iron or steel pipes, with a sufficient number of lengths of rubber piping to give flexibility to the apparatus. This enables the nozzle to be readily moved from point to point, and provides for the movement of the dredger itself, if working in rough water.

Choking of the pipes when working in sand is completely avoided by means of special vacuum and pressure-gauges and relief safety-valves working automatically.

When hard ground has to be dealt with, which the suction of the pump alone is unable to disturb, the difficulty is overcome either by an arrangement of revolving cutting points, driven by connecting rods from the engine, or by a special adaptation of a patent grab, which loosens and breaks up the material ready for the action of the pump.

The pump is fitted with a steam air ejector, by means of which it can be started in 30 seconds, and avoids the necessity of a foot-valve, which could not work in grit or stones; and by a novel arrangement of the shape of the delivery pipe it remains charged after stopping, ready for a new start.

The fan of the pump is driven at a high rate of speed, by a belt from the engine; this arrangement acting as a safety valve, as directly the pump shows a tendency to choke, the belt slips instead of the machine breaking.

The amount of work of which the Ball dredger is capable, and the moderate cost of same, may be estimated by the following extract from a paper read by A. A. Langley, Engineer-in-Chief of the Great Eastern Railway Company (on the work done by him at Lowestoft), at the Institute of Mechanical Engineers, London, in January 1882.

"The fan makes 350 revolutions per minute, and at that speed is capable of raising 400 tons of sand, gravel, and stones, per hour, but the average in actual work may be taken at 200 tons per hour. This is with a ten horse-power engine, and working in a depth of water varying from 7 to 25 ft.

"The figures for cost with this dredger are as follows :—

Cost of dredger, say	£2000
Nominal horse-power	10 H.P.
Depth at which it excavates	25 ft.
Quantity it excavated per annum	200,000 tons.
Labour for dredging, lead (2 miles) and discharging .. per ton	2'125d.
Coal and other stores.. .. .	0'375d.
Repairs (total £60)	0'072d.
Total working cost	2'572d.
Add 10 per cent. interest on capital	0'240d.
Total cost per ton, including interest on capital	* 2'812d.

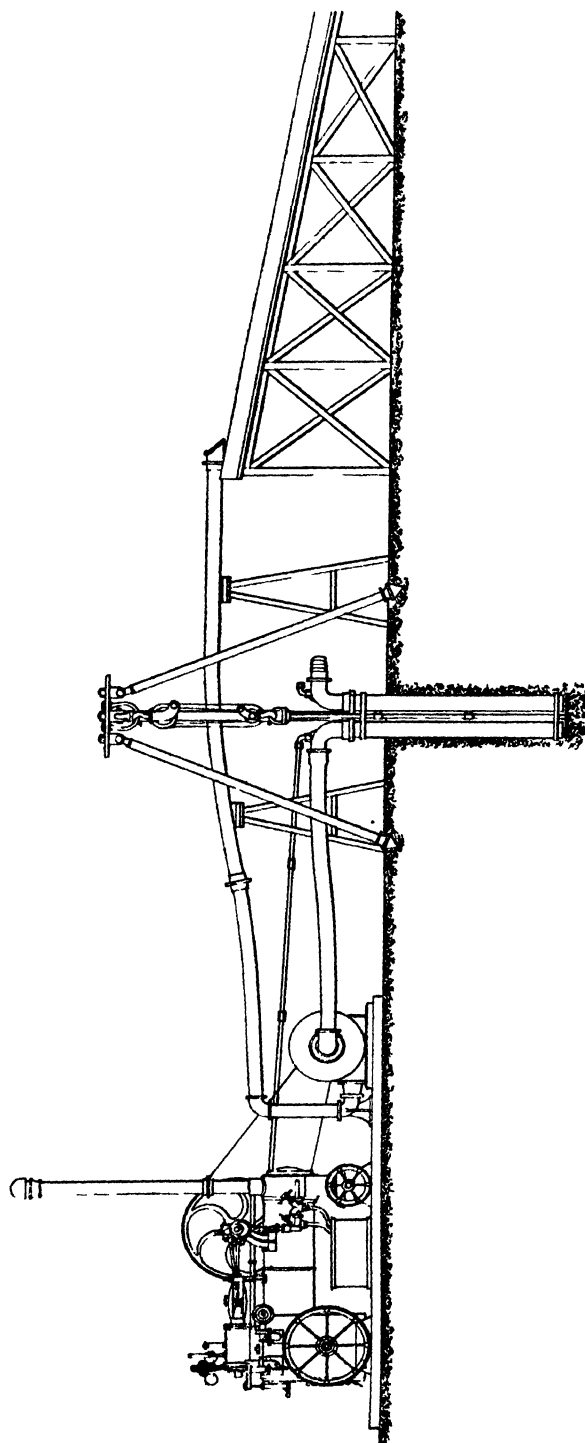
* This cost may be taken to be about 1½d. for dredging, and the balance for the towage of lighters and discharging at sea.

"The consumption of coal is about one ton per 1000 tons of sand dredged."

The dredger referred to in this extract had pipes 12 in. in diameter, and the material dredged was of a very favourable character, so that the quantity possible to be raised in other places would have to be reduced in proportion to the adverse nature of the material.

The following, however, may be taken as fair estimates of the capabilities of the pumps of various sizes, working in moderately loose stuff, the dredging machinery being either carried on a pontoon or barge afloat upon the water covering the material to be dredged, or fixed on land at

FIG. 42



THE BALL STRATUM TESTER AND SMALL GOLD DIGGERS' PUMP.

an equivalent height above the level of the water, the height of delivery being 10 ft. above this level.

Size of Piping.			Solid matter delivered per hour.			Water in addition, per hour.		
4 inch	..	..	About 10 tons			..	..	About 50 tons.
5 "	..	..	"	15 "	..	..	..	" 75 "
6 "	..	..	"	25 "	..	..	..	" 125 "
7 "	..	..	"	35 "	..	..	..	" 175 "
8 "	..	..	"	45 "	..	..	..	" 225 "
9 "	..	..	"	55 "	..	..	..	" 275 "
10 "	..	..	"	70 "	..	..	..	" 350 "
11 "	..	..	"	85 "	..	..	..	" 425 "
12 "	..	..	"	100 "	..	..	..	" 500 "

If the material were sand or loose gravel, about double the above quantities might be delivered.

A modification of Ball's dredger used so successfully in river-bed mining has been adapted to dealing with auriferous gravel beds, either for the purpose of proving the character of the deposit, or for actually working it. Such an arrangement is shown in Fig. 42. When sufficient water is not present in the deposit, a supply may be brought into the pit where the pump is working, with, if necessary, a further supply to the washing table. This has been adopted in some placers in Cape Colony and in New Zealand, and has given satisfaction.

Beach-mining or beach-combing.—On the Pacific shores of America, extending in patches for a considerable distance, are a number of auriferous deposits, known as "ocean-placers." Similar formations exist also in New Zealand. The beach sands contain minute scales of gold and platinum, and a large percentage of magnetic iron or black sand intermixed with the common beach sand, composed principally of quartz. The gold occurs in sufficient quantities to make mining for it remunerative at certain places. In beach-mining, considerable aid is derived from the natural causes at work—the winds, tides, and surf,—which act as a natural separator in parting the light and useless from the heavy and valuable particles. The force of the wind drives heavy swells upon the beach at high tide, and with them a certain amount of sandy matters; while at the ebb of the tide, the surf lashes the beach and carries back the light portions of the mass with the undertow, leaving some of the iron-sand, gold, and platinum, whose greater specific gravity enables them to resist the force. At low water, the miners go down on the beach and scrape up the iron-sand, which is generally left in thin layers, stacking it back from the reach of the surf, and subsequently washing out the gold. The position of the sands is liable to be changed at every tide, so that a fresh survey has to be made each day before commencing operations. When the richest spot has been chosen, mules laden with

empty *alforjas*, or raw-hide sacks, are led down to it at low water, and the most specious-looking portions are hastily gathered for treatment at leisure, a couple of days often sufficing to wash the harvest of a month.

For a long time, the common mode of washing was with a small stream of water in a sluice, terminating on an apron or tom, with turned up sides and ends, the bottom perforated with small holes, and underneath a wooden box set slightly sloping, and floored with a covering of amalgamated sheet copper. The auriferous sand is washed upon the apron, and, passing through the small holes, strikes vertically upon the copper below, the gold adhering to the copper, while the worthless sand passes off through an opening in the lower end of the box. This process is manifestly imperfect, as all the gold that does not instantly adhere to the copper is washed off and lost.

The ordinary blanket-sluice, or a combination of amalgamated copper plates, riffles, hanging-plates, &c., such as the Russel amalgamator, were for a long time relied on for catching the gold. But when the miners assayed their tailings, they found that in most cases fully $\frac{1}{2}$ of the gold had been lost in the treatment. Not only was the separation based on a difference of specific gravity quite unavailing, but examination showed that the gold-specks were coated with "rust," which prevented amalgamation. Experiments were then instituted on all hands. Chlorination and other chemical processes were found too expensive, but confirmed the richness of the sand, showing that much of it was worth 2*l.* to 6*l.* per ton, and some a great deal more. The quantity yielding 1*l.* to 2*l.* per ton is said to be practically inexhaustible. A company owning a large extent of these diggings is reported to have elaborated a process (Sublett's) capable of saving 90 per cent. of the fine or flour gold, and costing only 8*s.* to 12*s.* per ton, including mining, which is often only nominal. This process will be described under the head of "Fine gold" (see p. 231). Skidmore states that this machine "for saving gold from the ocean sands has never amounted to anything that I know of." He declares that the variable character of the sands prevents such an enterprise being carried on successfully for any length of time.

According to S. B. Christy there are alternating layers of magnetic iron sand and of common quartz sand upon the higher parts of the beach, while lower down, where the surf breaks, the sand becomes usually coarser and more full of pebbles. Layers of black sand are not infrequently found upon the top of the partly dried, undisturbed sand, owing to the removal of the lighter siliceous material by the action of the wind. As one digs down, the layers alternate white and black, the thickness of each varying from that of a knife-edge to 3 and 6 in. At length, at a depth of 2 to 4 ft., the "hard-pan" is reached. This hard-pan is composed of a sort of clay sandstone; upon it is usually found a layer

of coarser sand or gravel, which, especially when in depressions and crevices which favour a concentrating action of the waves, is usually the "best pay." But some of the higher layers of black sand are quite rich also. They seem to be formed upon a "false bottom," as the miners call it, by some high tide which makes a new deposit, while its waves are not violent enough to disturb the previous layers.

The concentrating action of the waves upon an inclined beach must be considerable. The configuration is entirely changed after a storm, and the whole area as far up as the cliffs is entirely covered by the surf during heavy weather. The action of the water on the sands is essentially the same as in the miner's pan. The quartz sand has a specific gravity of only 2.5, while that of the magnetic oxide of iron is usually about 5; hence the separation of these two materials is tolerably complete. The other heavy materials, as gold or platinum, naturally remain with the magnetic sand.

Under the microscope, the black sand shows itself to be composed of more or less perfectly rounded grains of magnetic oxide of iron, with a few silica particles, and occasional fragments of what appear to be garnets. Mixed with the black sand is the gold, which, under the microscope, is almost invariably bright, and in only a few cases presents a rusty appearance. It is always in the form of oblong or elliptical scales, which are doubly concave, and are usually surrounded by a thickened rim. A number measured by Clarke gave the following dimensions:—

Largest	{	0.0065 in.	Smallest	{	0.0030 in.
		0.0060 "			0.0030 "
		0.0070 "			0.0035 "
					0.0035 "

Measurements made by Christy gave the following results:—

Gold 0.0108 in., 0.0098 in., 0.0049 in.

Platinum occurs also, though not in large quantities. The fragments are usually more compact and less flattened than the gold particles. One specimen measured 0.0044 in.

As the particles of magnetic oxide are very nearly of the average diameter of the gold particles, and as, moreover, they are rounded, it becomes a matter of the greatest difficulty to separate them by washing, in spite of their great difference of specific gravities, which are as 5 to 19. The leaf-like form of the gold renders it practically as easily moved by water as the magnetic oxide. It is only by the skilful touch of an old miner that a good "colour" can be shown in the pan.

"These sands have doubtless been deposited by some ancient river flowing through the gold region of Oregon and California, which for ages

carried in suspension, in its current, immense quantities of auriferous gravel and which gradually filled its bed and the surrounding country with its deposits, and subsequently became elevated to its present position above the surrounding country.

"In some cases the whole length of the bluff is gold-bearing, with rich and poor streaks, yielding on the average 20 cents to the cub. yd., but the rich streaks giving as much as \$135 to the ton, of gold .957 fine. The sand contains gold and platinum in small quantities, in minute scales, most of which are flat, mixed with a very large percentage of black, magnetic, and ordinary quartz beach sand. The shape of the grains, which can be seen when they are highly magnified, is oblong and doubly convex, with edges thicker than the middle, a form which would naturally result from the continued pounding of the grains. Their thickness is rarely more than 0.001 of a line. In length they vary from 0.3 to 0.7 line.

"It was at first supposed that these beaches were confined exclusively to the coast-line, but it has since been discovered that older beaches containing the black sand exist as much as 30 miles back of the coast-line, and often at considerable elevations, and in such a position as to show that their elevation must have been very gradual. The black sand containing the gold, is sometimes covered to a depth of from 3 to 60 ft. with sterile material. In Oregon these are called "back beaches;" they are worked either by stripping or tunneling.

"All the early methods of extracting gold were based on the difference in specific gravity. The best of them did not save more than 20 per cent. of the gold. Plattner's process was tried, but was found to be too expensive to be used on a large scale, but the use of it for a short time confirmed the fact that fully 80 per cent. of all the gold contained in the sands was being lost. When these sands contain gold that can be seen by the microscope, most of it is quite bright, though it is covered occasionally by a coating which prevents the contact of the mercury. In most cases this appears to be of an organic nature, and seems to have been produced subsequently to the formation of the bluff.

"The remarkable purity of the gold obtained from the beaches has attracted a great deal of attention. This is undoubtedly owing to the fact that this gold being in an extremely finely divided state, and wet with the salt water for so long a time, has had the silver separated from it in the state of chloride, as is also the case with the other metals, leaving the gold almost pure. The value of these sands is very variable, depending on the situation and the state of the natural concentration at any particular time. Values of from \$10 to \$30 a ton are quite common, but the sand often has less, and sometimes contains many times this amount.

"As the operation of mining is entirely dependent for success on

the state of the weather, the direction of the wind, and the force of the tide, it is always uncertain, but as it requires very little outlay for plant, and not a great amount of labor, it is quite attractive; but as the concentration of the gold in sufficient quantities to work is dependent on the weather, the best claims are often unworkable for months, during which time the miner must wait patiently for any change of wind which will throw the sands up on the beach where he can reach them. He may not even be rewarded for many weeks for his patience in waiting by any days of favourable winds, or even by a single tide. The sea-water cannot be used without great loss for concentrating by hand, as it is too heavy, so that the beach miner must wait for nature to do the concentrating, which, when once effected and collected for treatment, may be entirely washed away in a very few minutes by a contrary wind or a heavy tide. Such uncertainties do not invite the investment of any large amount of capital.

"At Ophir Beach, near Rogue River, Oregon, the beach below high water is washed in a small way by the sluicing process, when the sand that is thrown up is rich enough to pay. The superintendent examines the beaches every day, collecting the sands and scraping up the loose gravel; he pans it, and on finding a "prospect," sends for a mule train, each mule carrying two coarse hide or canvas bags, which will contain about 125 lb. each; a train of 40 mules would, therefore, transport 10,000 lb., or 5 tons at a single trip. In order to collect the sand, the top gravel is stripped, and the black sand is gathered into piles. The canvas sacks are taken from the mules and filled, the mules being trained to start on a run for the works the moment they are loaded. As in some places the sea comes close up to the cliff, the animals are trained to turn their heads toward the cliff, bracing their feet firmly, allowing the wave to dash over them, and then going on their way on a run till the next wave comes. During a single tide, these mules under favourable conditions are able to make three trips, when the distance to the working place is not too great. At the works the sand is piled up and washed in rockers, long toms, or sluices, with amalgamated plates.

"At Randolph, in Oregon, at the mouth of Whisky Run, the gold in the sands was always very fine. No nuggets of more than 4 to 5 cents in value were found, but the sand was very rich. Only a small portion of the gold was saved. Much of the ground was worked over six or seven times in succession, and was each time found to be remunerative. For a very short time, a single miner could earn a thousand dollars a day, but the prosperity was of short duration. The town has been in ruins for many years. The largest amount ever taken out of any of the California beach-claims is said to have been \$25,000 in one year. The great difficulty in working the beaches has always been the lack of

fresh water. They have been washed with salt water pumped from the sea, but the amount of gold collected in such cases was only a small portion of what was lost.

"All the experience of these beach mines goes to show that immediately after a heavy cave or slide on the banks, the beaches are rich in coarse gold, but up to the present time it has not seemed profitable to do any bank-blasting, or to undertake mining on a large scale by the hydraulic processes. The gold is washed up by the sea only. Many suggestions have been made that the gold would be found in the sea beyond the surf-line, and some attempts to dredge for it have been made, but unsuccessfully.

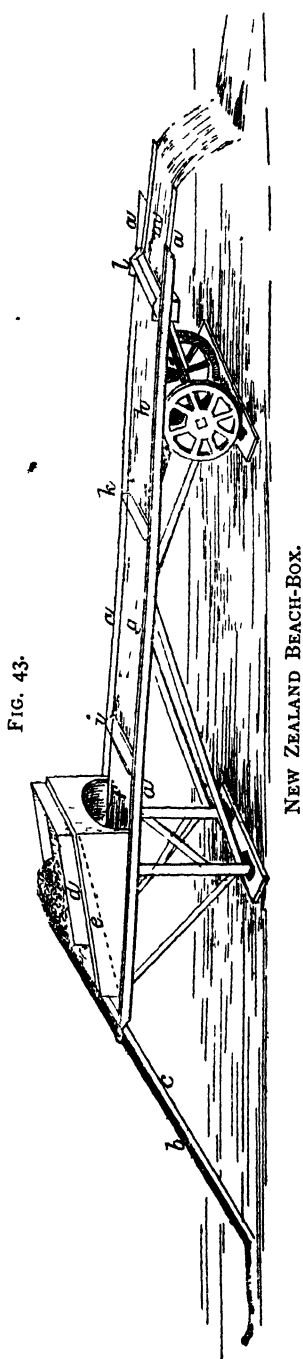
"It was proposed at one time to effect the amalgamation of these extremely light particles before endeavouring to concentrate them, by heating the sand and quicksilver under water up to the boiling point, which would cause the mercury to diffuse itself through the entire mass, and thus attack the gold that was in a proper condition to amalgamate. If this sand had been previously calcined, the first calcination and the subsequent heating would be sufficient in most cases to remove the coating upon the gold. In some cases a chemical treatment of the sand was made, and after all this was done, the fine particles of amalgam had still to be collected by boiling, and unless they were very rich and fuel plenty, it is evident that such a method as this, although patented and supposed to be worth a deal of money, could not be successful. The process consisted in screening the sand and subjecting it for 24 hours to a solution of caustic potash and salt, for removing any coating upon the gold, and destroying the compounds of sulphur which would attack the mercury. The material so treated was heated in a pan by a jet of steam, and the quicksilver is poured in; it was kept constantly in agitation by the steam, and the movement was kept up for $\frac{1}{4}$ to $\frac{1}{2}$ hour, by which time the gold should have become thoroughly amalgamated. The pulp was discharged into a vat to cool. It is desirable that it should not be heated too much, as that divides the quicksilver more than is necessary to secure the amalgamation. Care must be taken to use pure quicksilver in the process. The cooled material was then passed over a series of copper rollers grooved spirally and set one above the other. These are set over a sluice, 3 ft. wide, with amalgamated copper plates and grooved copper rolls. The rollers are hollow, 1 ft. in length, and 2 in. in diameter. As the amalgam accumulates on the rollers, it is either cleaned off or drops into the sluice below. Notwithstanding all that was hoped for it, the process was not successful." (Report of State Mineralogist).

In New Zealand a considerable amount of "beach-combing" is carried on where water is available, and with most satisfactory results. At North Beach, a head race was carried 5 miles from the Maori river,

costing 600*l.*, and the washing yielded 1 oz. a day per man. At Seventeen-mile Beach, near Greymouth, are over 10 miles of auriferous sea-shore, with at least 4 lines of pay-dirt, and extensive operations are now being carried on. About 2 miles north of Charleston is a settlement of Shetlanders, whose working time is divided between small farm cultivation and the gathering of fine gold from the sea-sands, which, after heavy weather, are easily got at and found to be highly auriferous. These beach-claims are deemed valuable property, and are kept carefully registered. Two double-area claims, each having 200 ft. frontage, with all working appliances, and a large water-race heading from the Little Totara river, fetched 1000*l.* in February, 1882. The claims alone are commonly sold for 200*l.* to 300*l.* each, independently of water-race and appliances.

The following account of beach-mining, as here conducted, has been very obligingly contributed by Warden Revell, in response to a special application on the subject.

Fig. 43 gives a sketch of a "beach-box," which is a fair representative of those now in use. The frame *a* is built entirely of timber, and from end to end measures 14 ft. The hind legs are 2 ft. high, and at the lower end the floor is 18 in. from the ground. The fall is regulated by raising the legs or wheels as may be necessary. The "hose" *b* is 3 in. in diameter, made of stout canvas—generally tarred to preserve it and render it watertight—through which the water is brought and conveyed up the 'leader' *c*, thence emptying into the hopper *d*. The bottom of the hopper is a sheet of perforated iron, 2 ft. sq.—the size of the hopper. The perforations are $\frac{1}{4}$ in. in diameter, and generally about 1 in. apart. The sand is shovelled into the hopper, and the finer part of it is carried through the perforations by the water, and drops upon the 'slide' *e* (shown on the sketch by dotted lines), within the box below the hopper.



Over this slide is laid a copper plate, coated with mercury, to which the gold attaches itself. This slide is 2 ft. wide and 3 ft. long; at its lower end an opening is left 2 in. from the plate beneath it, and the same space from the end of the box, so that the water may drop on the next plate *f*, and from that over plates *g* and *h*, which are also overlaid with mercury. At the bottom of plates *f g h*, "riffles" *i k l* are placed; *i* and *k* are formed by leaving a space between the plates, so as to form a hollow into which the water must drop before reaching the next plate. Above these two riffles, slides are placed to break the impetus of the water when it reaches them, and force it more gently under them. The last riffle-box is of different construction from those above. It is 5 in. in depth at the upper side, 13 in. on the lower, and 5 in. wide. In the centre a slide is placed, and pushed down to within $2\frac{1}{2}$ or 3 in. of the bottom of the riffle-box. The sand and water drop into this box, and are forced out again at the lower side of the "slides," thence running over the last table or "tail-board" *m*. The object of the two first riffles is to break the force of the water, and that of the riffle-box is to catch any loose mercury that may come off the plates above it, and which, on account of its weight, cannot be forced out again by the water. On the "tail-board," instead of mercury, "baize" or blanket is placed, to catch all "rusty" gold that the mercury will not take. On the lower side of each riffle, a "flange" is nailed so as to overlap the plate below and thus prevent the water passing beneath. The plates are secured to the frame by battens, and are nailed at top and bottom. Each plate usually weighs about 14 lb.

When sufficiently rich, the plates are taken from the frame and set on end, so that the loose mercury may run off. They are then scraped with a chisel, both gold and amalgam being taken off. After all the mercury possible has been squeezed from the amalgam, the latter is retorted in the usual manner to free the gold of the mercury entirely. Should any of the copper have been scraped off with the mercury, it will remain with the gold, which latter has then to undergo the process of refining at the banks.

The working results of these claims are various, and almost impossible to estimate, the beaches being often unworkable for months, for it is only heavy seas and strong winds that bring payable gold upon them. Revell thinks, however, that the average wages throughout the year would not be less than 2*l.* 10*s.* per week.

When, by reason of long duration of calm weather, the claims are rendered unworkable, the holders may obtain "protection" from the Warden for any period not exceeding 6 months. Many avail themselves of this privilege to cultivate the ground round about their cottages—their "Miner's Right" entitling them to hold 24 × 48 ft. without regis-

tration, or to the extent of an acre upon registration being granted them by the Warden. Fig. 44 shows a party of Shetlanders working the black sands on the Nine-mile beach, Charleston.

Many of the "beach-combers" are of opinion that the gold on their claims comes from the depths of the sea during the storms, and not from

FIG. 44



SHETLANDERS WORKING BEACH-DIGGINGS IN NEW ZEALAND.

the inland workings, as many others suppose. It is, however, a difficult question to determine.

Flume for transporting timber.—As a considerable quantity of timber is consumed in the construction of box-slucies and most other mining or washing implements and machinery, and the difficulty and cost of conveying the timber to the spot where it is required are often considerable, no apology is needed for inserting here a description of a flume used for carrying timber to the Virginia and Truckee railroad, in Nevada which might often be adopted with advantage.

It consists of a V-shaped launder or trough, made of 2-in. plank, 18 to 20 in. wide, supported by a light trestle-work, in which a rapidly flowing stream of water carries down cordwood or timber, of such lengths as the curves of the flume will allow passage to, from the head of the cañon to its mouth. In a flume of this description, light timber or plank 10 ft. in length is easily floated and transported. The average grade of the flume is 5° to 6° ; the minimum employed is 1 : 64, though that is

rather too slight; 1 : 33 or 3 ft. in 100 ft. is a very good grade with a fair volume of water. The flume referred to is 5 miles long: cordwood is said to make the entire journey in 18 minutes, and 51 cords of wood have been transported from the upper end of the flume to the place of delivery in 6 hours. At the end of the flume is an iron grating, having the reverse shape of the trough, and inclining upwards from the bottom to the upper edge of the flume. The water passes through this grating, while the wood shoots up on it and falls over the edge of the flume to the ground, where it is piled up or loaded upon the car. This simple method of handling wood cheapens its delivery very much. No men are required, except those who supply the wood to the flume at the upper end, and those who receive and stack it at the lower end. Along the course of the flume, feeding streams are brought in from side cañons at intervals of 1 to 2 miles, to increase the supply of water, and compensate for the wastage occasioned by leaks or over-running. In some of the steeper side cañons, it is possible to use "dry shoots," or similar flumes without water feeding the main flume, the inclination in their case being sufficient to enable the material to slide by the force of gravity alone. By this means, wood is easily delivered from points where roads are impracticable.

Cost of alluvial mining.—The following notes of the cost of alluvial mining in a variety of instances cannot fail to be interesting, as they afford the means of comparison for new undertakings. But the special circumstances surrounding each case must very largely govern the figures.

At Ballarat the average cost of raising and puddling wash-dirt and getting out the gold in several mining properties is 7s. 3d. per ton, and the cost of puddling and sluicing in two instances is 1s. 7½d. per ton.

At Clunes, the average cost in one alluvial mine is 10½d. per ton for puddling, and the average cost of sluicing is 3½d. per ton.

At Sandhurst, the average cost of raising "cement" is 3s. 6d. per ton; the average cost of carting, crushing, and extracting the gold is 8s. per ton; and the average cost of puddling or sluicing is 3s. per ton, including cartage.

At Maryborough, the average cost in three instances of raising cement is 21s. 8d. per ton; puddling, 2s. 6d. per ton.

At Castlemaine, the average cost, in one claim, of breaking cement is 21s. per ton; raising it to the surface and delivering it at the machine, 4s. 9d. per ton; crushing the cement and extracting the gold, 3s. 9d. per ton. The average cost of puddling by one party of miners is 1s. 2d. per ton.

At Maldon, the average cost, in one mine, of raising cement and delivering it at the machine, is 1s. 6½d. per ton; and the cost of crushing

and extracting the gold is 2s. per ton. At the Forty-foot lead, the average cost of raising, carting, and puddling is 6s. 6d. per ton.

At Hurdle Flat, Ovens district, ground 16 ft. 6in. deep, sluice in rock, 4 gr. of gold (6d. per load) pays well; 4 men get down and wash 1 ton of dirt every 5 minutes.

These figures all refer to Victoria, and are mostly taken from Brough Smyth.

Cement: its occurrence, treatment, and yield.—On nearly all alluvial gold-fields, whether shallow placers or deep leads, is found a stratum of ferruginous conglomerate, composed principally of rounded and angular fragments of quartz of all sizes, cemented (hence the name) together by the oxide of iron with which the mass is impregnated, and often so hard as to resist everything but blasting. It overlies the bed-rock, in some places resting on it, in others several inches or even feet above it. In thickness it fluctuates considerably, from 6 in. to 8 ft. or more. Its character varies but little. On the fields near the site of the present town of Maryborough, in Victoria, the depth of the cement was 10 to 13 ft., and it occupied the miners 2 to 3 weeks to get through it by the use of gads. The bottom was a soft, white pipe-clay, and though the wash-dirt taken out at that time was only 6 to 9 in. deep, the nuggets which could be picked out paid the men well. On another part of the same field, the depth was 16 to 24 ft., and the 4 to 6 in. of pay-dirt taken out was full of nuggets. A rush set in, and within 3 months from the date when the first prospector's tent was pitched on the site of Maryborough, at least 30,000 miners occupied the ground. At first, the cement was washed in a long-tom, which was, of course, quite unfitted for extracting the gold, and very much therefore was lost in the tailings. In some instances, a quantity of débris is found overlying the cement, and pays well for washing, as, though not so rich as the solid mass, it is more easily worked. When the miners realised the fact that the gold was imbedded in the cement in such a manner that mere washing would not dislodge it, they began to put it through the quartz-mills, with greatly improved results, and an immense amount of gold was got out of the tailings that had been previously cast aside. Where crushing-machinery was not at hand, the cement was often thrown out of the way, and huge heaps of it might be found about some of the older gold-fields.

Stamping.—The class of machinery first used for reducing cement was the ordinary stamp-battery employed in quartz-crushing; but an improved modification was adopted at some mills, viz. the holes in the screens were made $\frac{1}{4}$ in. instead of $\frac{1}{8}$ in. as usual, thus increasing the crushing capacity 25 per cent. without affecting the product of the cement. Warden Carew reports from the Tuapeka district of New

Zealand that quartz-stamps provided with a screen made of a kind of wire netting, with $\frac{1}{8}$ -in. meshes, as a substitute for the ordinary quartz screen of 120 holes to the in., are working very successfully on cement.

For a long time no other plan was tried, as this succeeded in liberating all the gold. Latterly, however, various devices have been proposed as cheaper substitutes for stamping.

A company owning many acres of rich cement, said to be 40 ft. thick, in Nevada, have brought a tunnel 2000 ft. long and 10 ft. wide with the intention of putting down two flumes throughout the entire length of the tunnel, and extending them as far beyond the mouth as may be desirable, there being plenty of fall for "dumps" (i. e. perpendicular drops of considerable height, where the concussion of the fall serves to break up the agglutinated masses) and under-currents. They calculate that by this means the cement, which has hitherto been crushed in stamp-mills, will be completely disintegrated, and the gold set free.

At Amelia, in Eastern Oregon, are cement beds of a known length of 10 or 12 miles, and varying in thickness from 10 in. to 10 ft. and upwards. They lie near the surface, and every rivulet cutting through them has yielded pay-dirt, and in some cases much riches. The bed-rock immediately beneath the cement has often yielded 4s. to 17. per pan. A short wet season induced the owners to try a novel method of treatment. During the winter, the cement was excavated and thrown up in heaps, where it slaked and disintegrated under the influence of alternate frost and thaw, and in the spring it was run through sluices, and gave very profitable results. Raymond expresses himself as "not certain that this method will not prove, after all, the best; though with a continuous supply of water throughout the larger portion of the year, it may not be necessary to confine the period of extraction to the winter, nor the period of working to a few weeks in the spring. Probably the disintegration of the cement could be effected at any season by allowing it to lie a certain time, and occasionally wetting it down."

Drake's cement-mill.—A new mill for reducing cement, known as Drake's cement-mill, has been made by Prescott, Scott & Co., Union Iron Works, San Francisco, for the New York and Calaveras Gold Co. It is in the form of a tube, 40 ft. long, the stakes of which are 5-in. T-rail bars. At the upper end of this tubular cage, the cement as broken from the mine is "dumped," or allowed to fall a considerable height from iron cars. When the machine is in motion, the gravitation of the cement drives it against the sharp edges of the T-bars, by which it is disintegrated. At the same time, a large body of water is allowed to enter the upper part of the tube. This helps the work of separation, and carries the finer material and gold between the bars to an apron beneath, which conveys it into the amalgamating-sluices. The boulders and coarse material slide gradually

down the machine, over the central and lower riffles, and out to the waste-heap. The machine weighs about 55 tons as it stands, and will be driven by a 50-H.P. engine, with a 14-in. $\times$ 24-in cylinder, and supplied with steam from a boiler 16 ft. long and 50 in. diameter.

Cox's pan.—Another apparatus, for reducing cement by grinding and friction instead of by crushing, is known as the Cox pan. The boulders, which rarely contain gold, are crushed under the stamps with great waste of time and power. This pan was invented to avoid the necessity for that step. It is about 5 ft. in diameter and 2 ft. high, and is intended to hold a charge of about $\frac{1}{2}$ ton. The rim is made of boiler-plate and the bottom of perforated cast iron, through which the finer sand and auriferous materials fall into the sluice-boxes, or other gold-saving appliances provided for their reception. The boulders and large pebbles are discharged at intervals through a section at the bottom of the pan, which is opened like a trap-door by means of a lever, when they have accumulated to such an extent as to retard the operations. Four revolving arms are attached to a shaft, which passes perpendicularly through the centre of the pan. On these arms are fastened steel teeth, something like plough-shares in appearance, which in the rapid revolution of the arms, break up the cement. An abundant supply of water is distributed while the pan is in motion, and materially aids in the disintegration.

At Table Mountain, in California, a company are using this pan for cement. The gravel is fed in continuously from a hopper, the feeding being interrupted only long enough to discharge the boulders. The dirt released by the action of the pan passes through the apertures in the bottom, whence it runs through 300 ft. of sluice-boxes. The proprietors speak highly of the pan after 2 years' constant use. At a neighbouring digging, two Cox pans are used, worked by 50 in. of water in an over-shot wheel 30 ft. in diameter. Each pan will treat 40 cart-loads = 40 tons per day. It is fed in continuously, about 10 in. of gravel being kept constantly in the bottom of the pan, which is charged with a little mercury while 100 to 200 ft. of sluice are used below the pan. D. M. Hughes the manager at this mine, has used the pan for several years, and effected some improvements in it, especially in distributing the water through the pulp while in motion. He says it will work any cement soft enough to yield to the pick, and work it better than any other process; but he admits the loss of a considerable percentage of the fine gold—which, it may be remarked, is not necessarily attributable to the pan.

Raymond gives some further particulars, furnished by another miner at Dutch Flat, where the cement is that known as "blue," and considered as hard as any in the state. The cement is dumped or tipped into a

hopper with an inclined bottom, whence it is loaded into the pan by a gate worked by the man in charge of the pan, thus requiring only one handling, which is a great saving of cost. About 1000 to 1200 lb. are usually put into the pan at once, while it is in motion. In the top of the rim of the pan is constantly a stream of 4 to 5 in. of water, which carries the pulverised cement down through the small openings of the bottom of the pan into the sluices, &c., below. The pan is set in motion, and the gate of the hopper is raised to gradually let the charge of cement enter the pan, generally completed in about 2 minutes. It is run about 4 minutes longer, then stopped to open the trap-door, and then again set in motion, so that all the boulders and stones are driven out and fall into a rock-sluice, the fine auriferous matters having already passed through the bottom of the pan. The operation is then repeated as before. Softer material requires less time. The pan will readily work 100 or 125 tons (of 20 cub. ft. to the ton) per 24 hours, and at a cost, including water-power, labour, &c., of about 5*d.* per ton. The hardest cement requires the pan to be worked at 65 revolutions per minute, consuming 8 H.P. This particular one is run by a "hurdy-gurdy" wheel, 10 ft. in diameter. Those previously mentioned are driven by similar wheels; but as the cement is not so hard in their case, the cost is 3½*d.* to 4½*d.* per ton. It is said to do its work very thoroughly and cheaply, every stone being perfectly cleaned, and the cement so pulverised that it is difficult to find the colour of gold in the tailings. The inventor claims that one pan will do the work of a 25-stamp mill. Its cost is small compared with that of the latter, being about 240*l.* The wear and tear is estimated at 5*d.* per day. The cement can be worked for about ⅓ the expense entailed by stamping, which costs 4*s.* to 7*s.* per ton, and is not so efficient.

Prof. Skidmore says that the Cox pan works well under certain conditions, but those conditions are rarely found. It is now nearly obsolete; the only two known to be in use have been erected at the Derbec mine near North Bloomfield, Nevada co., of which mine, the inventor of the pan, J. B. Cox, is superintendent.

Cement yields.—The yields obtained from cement are of course various, as in all classes of mining. At once place, 4 men crushed 4 loads of cement and got 184 oz. of gold, and another party 63 oz. from 15 loads. The wash-dirt with which the cement was intermixed was highly auriferous, and gold was found throughout from the surface to the cement encrusted on the bottom-rock. Other Victorian diggings have given:—5 dwt.; over 14 dwt.; 6 dwt. to ½ oz.; 12 oz.; 1 to 2 oz.; 5 to 8 dwt.; 4 oz.; 6 oz.; 7 oz.; 8 oz.; and 2 to 8 oz.—per ton; 5 tons gave 3 oz. 5 dwt.; 20 tons gave 360 oz.; 2 oz. per load were sometimes obtained; also 5 dwt. per load, much being lost; ½ to 1 oz. per load, much being lost; and 4 dwt.

per ton, when 130 lb. of tailings gave 17 gr. One of the wardens on the Otago gold-field, New Zealand, states that in the Tuapeka district "a single pennyweight of gold to a cubic yard of conglomerate (cement) will leave a handsome margin of profit." The average yield from all Victorian cement raised to the end of 1887 was 4 dwt. 15·45 gr., on an output of 380,388 tons. In 1887, 6569 tons of cement crushed afforded 1368 oz. of gold, or an average of 4 dwt. 3·99 gr. per ton; the highest yield was 1 oz. per ton from 7 tons in the Gippsland district, and the lowest was 2 dwt. 11·94 gr. from 3211 tons in the Maryborough district.

Saving fine, flour-, and float-gold.—In adopting measures for saving very fine gold, it is necessary to remember not only that the specific gravity of a grain determines its tendency to subside, but that also its size, shape, and affinity for other bodies must be taken into consideration. Iron is specifically far heavier than quartz, yet fine iron-filings will float on water while the finest quartz-sand will sink. Gold has a still greater specific gravity than iron, yet if fine gold and fine quartz-sand be placed in water, many particles of the former will be seen floating while the sand sinks at once. Observers have remarked that all metals seem to have a tendency to float on water when in a very fine state, though not always in the same degree. One way of accounting for this phenomenon is by supposing that minute globules of air or vapour attach themselves to the grain of metal, and thus overcome the force of gravitation. It is even asserted that this property or affinity is most strongly developed in the precious metals, while in compounds it is altogether wanting. Experiments have shown that fine particles of gold would remain afloat in water of ordinary temperature for over 24 hours, but that when the temperature was raised to boiling-point, or nearly so, the suspended particles of metal sank. One train of reasoning deduced from this is that the heat causes the bubbles of vapour to expand and escape. Another theorist denies that the "film of air which may adhere to the surface of a particle of metal has very much to do with its buoyancy in water," and prefers to seek an explanation on the ground that the "metals, when in small particles, are more or less in the state of *laminae*," this shape preventing them from overcoming the cohesion of the atoms of the fluid. Thus they remain suspended until some disturbance of the liquid turns them edgewise, when they sink, till some cause turns them flat again. The effect produced by heating the water is also ascribed to the expansion of the water atoms lessening their cohesion, wherefore they more readily yield to the grains of metal; the commotion also would constantly change the position of the grains.

Water.—Brough Smyth gives a practical illustration of the well-established fact that brackish or salt water, such as is frequently pumped out of mines, is quite unfitted for gold-washing. He remarks of the

Chinese washing at Sandhurst, in Victoria, that "in ordinary cases, the waters of the creek enable them to pursue their labours profitably ; but where even *clear* water pumped out of the mines is diverted into the creek, they are obliged to seek for supplies elsewhere. With rain-water, they can recover very fine gold ; but with brackish water, they get from a fourth to a third less." Smyth is disposed to think that the increased specific gravity of the water may partly account for this, that of the "water got out of the claims at Sandhurst" being "certainly not much below that of sea-water," and "Dr. Schweitzer's examinations of the water of the British Channel, show that the specific gravity is 1.0274 at 60° F." It is also reasonable to suppose that water holding various salts in solution is more likely to impart a greasiness to the surface of the gold particles, hindering their amalgamation, and perhaps facilitating the adhesion of those air globules which have an important effect on the floatability of the minute metallic atoms.

Copper plates.—On p. 222 some details have been given of the amalgamated copper plates specially devised to save the fine gold. These same plates, which have been the subject of much praise in America, were tried in Australia, and proved—by assays of the tailings—to be far inferior to the blankets commonly in use. Under the subject of Beach-combing (p. 216) are some remarks as to the inefficiency of the plates employed in that branch of gold-mining in California, yet with some modifications these plates are still in vogue. Grease or resin in the water greatly interferes with the process ; so also does a low temperature. It is said that a solution of cyanide or prussiate of potash is much better than nitric acid for use in applying the mercury to the plates, as there is then no trouble with the green spots of nitrate of copper. Küstel gives another mode of amalgamating the copper plates. He says "a good result is obtained by putting the copper plates into a wooden vessel with sufficient water to cover the uppermost plate 2 or 3 in. Very little sulphuric acid is added to the water—so much as to make it taste like strong vinegar. After 6 to 12 hours, the plates are taken out, washed in the same liquid, and the mercury is rubbed over the surface before the plate becomes dry. Washed in cold water, the copper plate is then prepared for use."

McDougall's plan.—McDougall, some years since, patented in America a machine for saving fine gold, which was employed for several months at least in extracting gold from the waste-water of the Eurcka and Idaho mines. It consists of 6 troughs, each 12 ft. by $2\frac{1}{2}$ ft., inclined at a slight angle, and having the bottoms covered with copper plates, amalgamated, and thickly studded with square iron pegs, about 4 in. high and $\frac{1}{2}$ in. square. Over these pegs, close-fitting copper caps, whose outer surface is amalgamated, are placed in such a manner that their corners

are presented to the stream. The waste-water, from which has been extracted all the gold that blankets, copper plates, rubbers, pans, riffles, buddles, &c., were capable of getting, is brought into McDougall's works, and runs through the troughs just described. Striking against the pegs, of which the 6 troughs contain 5000, the water surges and eddies about so that every atom comes into contact with the amalgamated surfaces. The precipitation of the gold is said to be greatly increased by the "electrical action induced by the differences in latent heat between the different metals, copper, iron, and quicksilver. Amalgam forms rapidly, and 2 men are kept constantly employed in cleaning the copper caps and plates. Owing to the almost microscopical fineness of the gold particles thus saved, the amalgam obtained does not contain as much gold to the ounce as that ordinarily obtained at the quartz-mills. This is, of course, to be expected. McDougall can tell almost instantly what grade of ore is being worked at the mills above him. When they are running what they call 'poor' rock, his contrivance saves the most gold; when they are crushing rich rock, he does not do as well. The explanation is, that the rock which they call poor may contain as much gold as the rich rock, but it exists in such very fine particles that their mill process cannot arrest it. It is these fine particles that he saves. In their rich rock, their gold being coarse, they save a greater proportion of it." (Raymond.)

Professors Raymond and Skidmore write at a subsequent date that McDougall's machine realised a good living for the owner for about 2 years. When the rock of the Eureka got poor, McDougall sold his invention to the company for \$700. It paid the company well for a few months, but was discarded when the ores ran to low grade. It may be considered a success when the ore is rich and the loss great. The invention is practically defunct.

Sublett's plan.—The process alluded to on p. 216, under the head "Beach-combing," is as follows. The sand is first screened in order to reduce its bulk, and is then subjected for about 24 hours to a solution of caustic potash and common salt. This is to remove any oxide or film that may be upon the gold, and to destroy sulphur and other base substances which would spoil the mercury. The mass is then placed in a pan, and constantly stirred while being heated by a jet of steam for a few minutes. The mercury is then added, and the steam and agitation again applied for 15 to 30 minutes, when the gold becomes thoroughly amalgamated, and the mass, or "pulp," as it is called, is discharged into a vat to cool before putting it through the separating sluice. The heat expands or partially evaporates the mercury, and, assisted by the agitation, distributes it through the mass in minute globules, where it meets and amalgamates with the equally fine and widely disseminated grains

of gold. Too great a heat must be avoided, as liable to flog the mercury more than is necessary, and thus increase the difficulty of re-collecting it. Considering that the amount of heat required to raise the temperature of 1 lb. of water from 32° to 212° F. will raise that of about 30 lb. of mercury through the same range, the reason for this caution is obvious. Great care must be taken to purify the mercury each time it is used, as success depends much upon this. Retorting alone is not always sufficient, as some of the base metals volatilise and pass over with the mercury. The most difficult part of the process is that of separating the fine particles of amalgam and mercury from the heavy black sand, for the plan adopted with quartz-"pulp" is not available for the heavier black sand. The valuable particles will not be precipitated by any concentrating motion, on account of their lightness and minuteness as compared with the magnetic oxide. The difficulty is said to be overcome, and the separation effected without material loss, by means of a system of galvanised copper rollers, grooved spirally, and placed side by side and in layers one above another, so as to break joints and not quite touch, and extending across the sluice, which is 3 ft. or more wide. A screen is placed over the rollers, to distribute the sand and water as they fall, and a galvanised copper plate is put beneath, to catch the mercury as it drips from the rollers. Two or three tiers of these rollers are thus arranged at two or more places, a few feet apart in the sluice, and drop-riffles or wells are sunk across the bottom, a little below the copper plates, to receive and retain the mercury as it runs from them. The rollers are 1 ft. long, 1½ in. in diameter, and hollow; 6 or 8 are laid side by side and end to end, extending across the width of the sluice. These and the copper-plates are kept in a highly sensitised condition and free from verdigris, and as the pulp passes down, over and between their multiplied surfaces, it necessarily brings the fine particles of mercury into contact with some one of them, to which they will adhere, before passing through the whole of them as arranged in the sluice. As the mercury accumulates upon the rollers, it drops from the under side, and the amalgam is cleaned from them in the same manner as from the plates. These amalgamating rollers and their application both to the cradle and the sluice are the invention of William Sublett, who has applied for a patent for them.

Flycatching.—It has been found at Charleston, New Zealand, that the gold does not all settle in the tail-race, but that on the union of the waters of several tail-races a small percentage, well worth saving, floats away. This is arrested by a method termed "flycatching," which consists of a series of blanket-tables placed across stream, like weirs, so that the waters shall flow over each table in succession. The tables are washed in turn, and the gold is streamered from the sand, and caught up by mercury. Many of these "claims" are very valuable, yielding 4/.

to 7*l.* a week with little labour. In the Charleston district, flycatching has become quite an industry in itself; the following description and accompanying illustrations are due to the kindness of Warden Revell, of that district.

The tables are constructed entirely of timber. Piles 2 or 3 ft. in length are driven firmly into the bed of the creek, and on these are nailed lengths of stout quartering, covered over with 1-in. boards laid close together, so as to form a smooth table. Pieces of lighter quartering are then placed over the boards from top to bottom, forming divisions about 4 ft. in width. Blanketing or cloth—ordinary corn-sacks opened out are frequently used,—is next spread smoothly along these divisions,

FIG. 45.



"FLYCA1CHING" TABLLS.

and securely fastened down by small strips of wood. The tables vary in length from 7 to 12 ft., and are placed in the creeks at intervals of 60 to 100 ft., extending quite across the stream. The proprietors of these rights realise during rainy weather very good returns, varying from 2*l.* to 6*l.* a week, according to the nature of the workings on the terraces above, and the number of tables set in the creek. The tables are costly and liable to be damaged by floods, and not unfrequently by cattle crossing the creek.

Fig. 45 shows one set of tables as set in the creek, with the water running over. Fig. 46 is a view of the tables in full work. The owner has turned off the water from one compartment prior to lifting the

blankets for washing out. Spare cloths are kept to replace those lifted, which, when washed out, are ready to fill the places of those in the next compartment. The men wash out the cloths in a large zinc box at the side of the creek. The cloths are generally washed out once a day. The fine tailings pass over several sets of tables in their course down the

FIG 46



LIFTING THE BLANKETS FROM "FLYCATCHING" TABLES.

creek. In the illustrations, there are about 12 sets of tables belonging to one man, and some 5 or 6 proprietors of similar rights occupy positions farther up the stream. The creek is named Darkie's Creek, and empties into the Nile river about 10 chains from where the man is standing in Fig. 46.

Coated Gold.—If perfectly clean gold is exposed to the action of pure mercury, it is instantly seized by the latter and coated with amalgam on the surface, but it is not penetrated or dissolved to any great depth. The fact of native gold being alloyed with other metals does not impair its affinity for mercury, if the surface is made bright mechanically by filing or scraping. But much native gold found in placer mines, apparently clean, is slightly tarnished by the oxidising or mineralising of its alloy, in which case it amalgamates with difficulty. It is very rare to find quartz gold in this condition. A large proportion of the placer gold found in California is wholly or partly coated with silica, cemented by sesquioxide of iron. When wholly coated, it is perfectly

inert to the action of mercury ; when partly coated, the exposed parts become amalgamated, and to that extent only is the gold held by the mercury. If this so-called "rusty" gold is digested in hydrochloric acid, the iron is dissolved, and a slight mechanical force then serves to detach the silica, when amalgamation takes place without difficulty. There is no hope of being able to free placer gold from this coating during the few hours it is exposed to the forces employed in the hydraulic process. Amalgamated gold could not, in the process of placer washing, escape from the mercury, but coated gold, under the same circumstances, will float on the surface of the mercury, and any slight force sufficient to overcome its specific gravity will detach it. The coating on placer gold seems to be a result of the pounding which it has undergone, forcing minute grains of quartz (silica) into its yielding surface, aided by the cementing action of sesquioxide of iron resulting from decomposed iron pyrites, which always occur in the quartz veins, whence the placer gold was derived.

Another kind of coating, which is equally powerful in preventing amalgamation, is due to a kind of mineral oil permeating the alluvial formation which contains the gold. This occurs in the Transvaal, and is a source of much loss.

It is difficult in the extreme to provide remedies for this coating which will not in materials or in labour cost so much as to be prohibitive. There is a probability that in some cases, at any rate, much good might result from opening up the gravel deposit and exposing it to atmospheric influences for a considerable period before commencing to wash out the gold. If, in the meantime, it could be well watered, so much the better, as that would induce a movement of the particles, and cause a certain amount of attrition. The water applied must be free from salts in solution, i. e. must contain no mine drainage.

Yields of shallow placers.—The following figures will convey some idea of the amounts of gold which have been obtained from various surface-washings in different parts of the world. In Victoria, at Dirty Dick's Gully, the miners worked all the alluvium 6 in. to 6 ft. in depth, and got 10 to 16 oz. per tub of 4 small buckets, while there were pockets containing 100 oz. and more. In Donkey Gully, the yield at the sides was 4 to 6 oz. per tub, in the middle 12 to 20 oz. ; within a length of about $\frac{3}{4}$ mile, more than a ton of gold has been taken from this gully. At Red Hill, the average was 12 oz. per tub, while 600 oz. were found in one pocket, and 30,000*l.* worth was extracted from a single claim. At another pocket, 1800 oz. were discovered. In Golden Gully, 18 to 20 oz. to the tub were common ; and 456 oz. were gathered from one hole. On Dinah Flat, 252 oz. were taken from one bucketful of dirt. At some places, pounds' weight of gold were not uncommon ; 4068 oz. were taken

by one party from a claim measuring 18 ft. by 12 ft., and some of their neighbours did even better. At Hundredweight Hill, as much as 360 oz. were washed out of one tub of surface soil. On workings 8 ft. deep at Forty-foot Hill, a claim 8 ft. sq. usually gave 3600 oz. of gold. Even with that rude implement the puddling-machine, 4 men at Castlemaine got gold at the rate of 100*l.* per week. At Sandhurst, the average amount of gold left in the tailings from puddling-machines was about 2 dwt. per ton. One man got 5000*l.* gold in 9 days. Nuggets of 22 lb., 108 oz., and 99 oz. are recorded. At Ballarat East, a tin-dishful of dirt taken from the surface of a hill gave as much as 168 oz. of gold. These figures are quoted from Brough Smyth, and are absolutely reliable.

In the Westport district of New Zealand, a party of 6 men got 800 oz. in 4 months; another party of 3, 100 oz. in 14 weeks; and a third party of 6, 140 oz. in the same time.

From California, come similar figures. One company took out 20,000*l.* per season for two or three seasons. A panful of bar gravel sometimes contained 20*l.* worth of scale gold. The average yield of the companies working on bars in one district was about 20 oz. a day. Individuals have been known to make 6000*l.* a season. A man with a rocker would make 200*l.* to 300*l.* per season. Many companies at Rose's Bar made 15,000*l.* per season in the rivers. At the mouth of the Rabb Ravine, near Timbuctoo, the yield was 20*l.* a day per man; there were 20 men at Timbuctoo who made an average total of 80*l.* per diem, without any proper facilities. On the Cement Mine Ravine, 2 oz. a day to the man was very common. At Square Creek Ravine, some companies made runs of 18 days, in which 3 or 4 men usually netted 1000*l.* to 1200*l.* At other spots, surfacers made 10*l.* to 20*l.* a day to the hand. One authority estimates the average yield of placers over a large area at a minimum of 2*l.* per diem per man, and thinks 3*l.* more nearly correct. Some individuals made 6000*l.* to 8000*l.*, and then left. These accounts show only the bright side of the picture, and it is scarcely necessary to remark that there have been instances where the cost of working was not even repaid by the gold found.

The average yield of all alluvial washdirt, excluding cement, raised in Victoria to the end of 1887, was 1 dwt. 11·79 gr., on an output of 13,392,842 tons. In 1887, 836,009 tons of washdirt puddled or sluiced yielded 82,513 oz., or 1 dwt. 23·37 gr. per ton. The highest return was 4 dwt. 13·55 gr. from 610 tons in the Ararat district, and the lowest was 23·75 gr. from 208,993 tons in the Castlemaine district. In New South Wales, in 1880, the average result of dealing with 292,184 tons of washdirt was 1 dwt. 2·73 gr. per ton, the highest yield being 6 dwt. 16 gr. from 3090 tons in the Mudjee district, and the lowest 4·34 gr. from 230,790 tons in the Southern district.

CHAPTER IV.

WORKING DEEP-LYING GRAVEL DEPOSITS AND THE ACCUMULATIONS
LEFT BY RIVERS LONG SINCE EXTINCT.

Definition.—A “deep lead” is a deposit of auriferous gravel, lying at a considerable depth beneath the surface, and often covered by beds of lava or basalt, hundreds and even thousands of feet thick. This deposit is to be distinguished from a shallow placer or surface washing, by the fact that it has been the work of a drainage system which no longer exists. Like the modern placer diggings, deep leads are generally, though not always, fluvial or riverine.* In California, indeed, they are known as “dead rivers,” a term which is applied only to channels which were occupied by running streams in past geological ages, and are now filled up with earthy or rocky matter. They are not to be confounded with channels that are open and remain dry during a part of the year from lack of water, or which have been abandoned by their streams for other channels. A dry river-bed is not a dead river. In almost all instances, these ancient deposits have been discovered by following up a modern auriferous stream, and tracing the wash-dirt onwards into the deeper ground, whence part of it had been dislodged by the recent stream. It is often easy to judge whether the gold in a modern placer or shallow washing has been derived directly from a quartz reef or from a deep lead, as that from the reefs is generally coarse, heavy, and not much water-worn, while that from the leads is usually fine and rounded.

The erosion due to rivers, and, perhaps, also sea-waves, has destroyed the veins, and formed a great deposit of mud and clay in the valleys, while in many cases the gold-bearing veins, the streams which wore them away, and the deposit formed by the streams, have all disappeared, and the gold has to be sought in more recent accumulations, which have undergone a variety of changes and perturbations.

It is scarcely possible to do more than guess at the age of the Tertiary rocks in which the deep leads are found, and the evidence on this head has to be sought rather in the relations borne by these deposits to

* An exception is to be found in the lacustrine deposits of the Otago gold-field, New Zealand.

the adjacent formations than in any fossils which they may yield. Still the limits are often sufficiently well marked to meet practical demands. The deep leads were long supposed to be in no case older than Pliocene ; no marine relic has apparently been found in auriferous drifts, and no gold-mining is known to be carried on in drifts underlying marine fossiliferous strata. But the greatest distrust must be maintained towards these dogmas ; the deep leads and other strata have only been very imperfectly studied, and at any moment gold may be discovered in Secondary or Tertiary fossiliferous rocks. Indeed, in the Fifth Report of Progress of the Geological Survey of Victoria it is claimed that extensive and rich gold-washings have been found and partly worked in distinctly fossiliferous Miocene strata, overlaid by a capping of basalt, in some places 700 ft. thick.

Rarely is it possible to trace the course of a deep lead within such limits as will serve the turn of the gold-miner, merely by the indications furnished by the natural surface of the land overlying them. The course of a channel originating in a range of Palæozoic rock may for some distance be well marked by strata of gravel, sand, and water-worn quartz pebbles filling up the lower part of the valley ; but when it is followed out into the low plains, through which the modern stream has cut its way, the strata may, and almost certainly will, increase in thickness, so that it becomes impossible any longer to follow the trend of the ancient river-bed. By noting the direction of the boundary ridges, it may be possible to indicate the limits of the deviations of the channel ; but where the distance between them is considerable, which is very generally the case, much money may be fruitlessly spent in sinking in search of the lead. In Victoria, some little guidance may be derived from the fact that it has hitherto been found that almost all the leads run parallel to the modern rivers, whereas in California they are more commonly at right angles to the latter ; and even under the best conditions, the sinuosities of the ancient channel cannot be taken into calculation, neither is it possible to foretell where the bed may widen and where it may become narrow, points which, as will be seen presently, have a vital bearing upon the payable character or richness of the auriferous lead. When a lead is overlaid by basalt, the miner's difficulties are increased. In some localities, it is true, the direction can be determined within limits sufficiently restricted for theoretical purposes, because the outcrops of the bed-rocks are visible on either hand, and their elevation above the valley or plain can be fixed. But to the miner, only two courses present themselves : if he will ascertain the course of the lead with accuracy, he must undertake expensive subterranean explorations ; approximately, he may succeed by putting down borings.

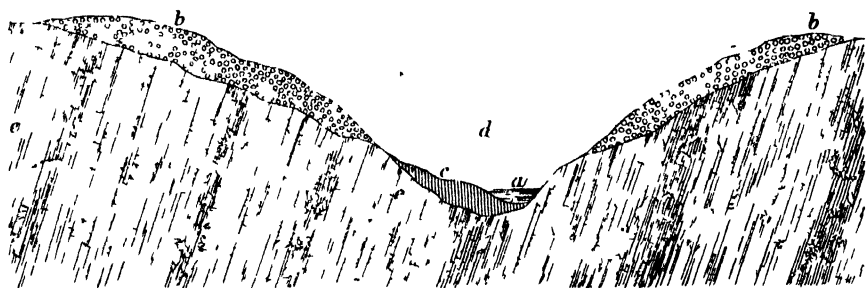
Yet even when the general trend is known, and the width reaches a

mile or two, nothing is easier than to miss the richest part of the lead. The undulations of the surface afford no clue, and are often very misleading ; and when the lowest depressions of the underlying strata have been found at the expense of boring, it may chance that the richest deposits of the gold do not lie there, but in the "reef-washes" or "benches" at the higher levels. A "reef-wash" is a deposit of wash-dirt spread over an expanse of flat or undulating "reef" or bed-rock, or lodged in a hollow in it above the level of the "gutter" or true bed of the water-course. For instance, towards the junction of two leads, the wall of reef or bed-rock by which the two leads are separated appears in some cases to be worn or broken down, and a stretch of comparatively flat rock is left between them : if covered by a layer of wash-dirt, and raised above the level of the water-courses, this would be called a "reef-wash." Also where the rock on one side of the water-course spreads out at a higher level, and a channel containing wash-dirt is found running alongside the true gutter, and between the gutter and the wall of the bed-rock, it is called "reef-wash." Several instances have occurred in Victoria, where considerable alluvial gold deposits have been found and worked at distances of 200 and 300 yd. from the gutter, which had already been exhausted and deserted. Sometimes the outcrops of deep leads may be discovered by prospecting along the edges of the basalts. It seems to be considered as an established fact that there are no auriferous quartz veins in the mesozoic strata of Victoria, and therefore it may be assumed that any gold found resting upon them must have been conveyed from older rocks containing auriferous matrices ; this is considered to preclude the probability of any rich deposit of gold being found on them.

Formation.—Auriferous leads seem, so far as is known, to present closely similar features wherever they may be found, whether exposed, or covered by basalt or recent sedimentary deposits. It must be remembered that even the hardest rocks are worn away by the combined action of air, rain, wind, sun, dew, and frost, and are rendered capable of transport by streams. These latter act in two ways : they cut into the rocks through which they flow, and at the same time they deposit in their courses the heavier particles of material brought down from the neighbouring elevations. In this way, alluvions are formed. If the stream is powerful, the fall not very great, and the rocks soft, a comparatively wide deposit of drifts and clays will be arranged, according to the specific gravity of the materials, modified, however, by the size and shape of the particles ; and these strata will gradually increase in thickness till the lower part of the stream begins to acquire greater force from the steeper fall there produced, and scoops out a deeper channel for itself. In time it will wear back towards the source, and a new and deeper course will be cut through the alluvions and bed-rock. The

action is illustrated in Fig. 47. The stream *a* represents the "gutter" of a lead; the drifts *b*, the "reef-wash"; *c*, newer drift; *d*, older drift cut through and carried away; *e*, bed-rock. It is often supposed that the reef-wash must necessarily be younger than the drift in the gutter, but it is quite possible that it may even be older in some instances. Evidently many successive layers of different ages may be formed in the same way, and where it can be proved that the stream has always been cutting a deeper channel, it may be assumed with certainty that

FIG. 47.



SECTION SHOWING "REEF-WASH."

the upper drifts or reef-washes are older than those lying deeper down. A stream will seldom continue to make a deposit in the direction of its course for a lengthened period and until its channel is changed, but this does occasionally happen where the bounding ridges are low. In such a case, the bed will gradually be silted up, and it will be almost impossible to ascertain the relative ages of the deposits, especially when the lithological features remain the same throughout. Reef washes are frequently known as "terraces."

The rate at which alterations are effected by streams of water depends in a great measure upon the nature of the rocks exposed to their action. When the strata are inclined, and composed of soft sandstones and argillaceous rocks, materials will be transported and new channels excavated in a manner which is not known when the rocks are hard or lie almost horizontally. The stream *a*, shown in Fig. 48, would rapidly erode the rock against which it impinges, and would be assisted by every shower of rain, the upper portions falling down as fast as the lower parts were worn away. The bed would constantly move towards the right and the earthy materials of all kinds would be disintegrated and transported to form a new deposit in some portion of the bed of the stream.

In Fig. 49 may be observed the effects of another co-operation of forces. The stream *a*, wearing away the rocks on the right, would cause masses to fall from time to time, and landslips would also take place.

Surface water percolating through the layers of clay at *c* would facilitate this, and an insignificant rivulet would soon materially change its bed. When claystones form a part of the strata in such cases, the layers will not hold together, and large masses will slide down at a time.

This kind of erosion often gives rise to important changes, of which a common example is shown in Fig. 50. Sometimes, by examining an ordinary horseshoe bend, it will be seen that the stream has at various times overflowed and denuded nearly every part of the surface of the peninsula *c*, which thus becomes much lower than the adjacent country.

FIG. 48.

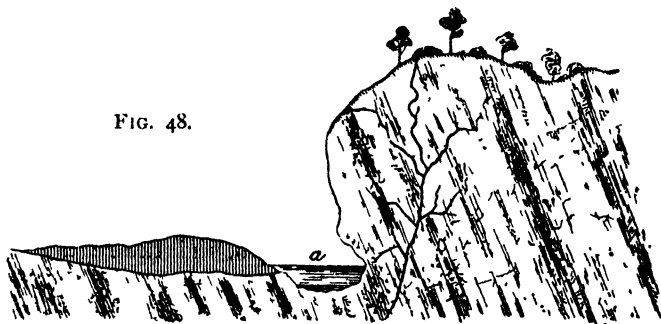
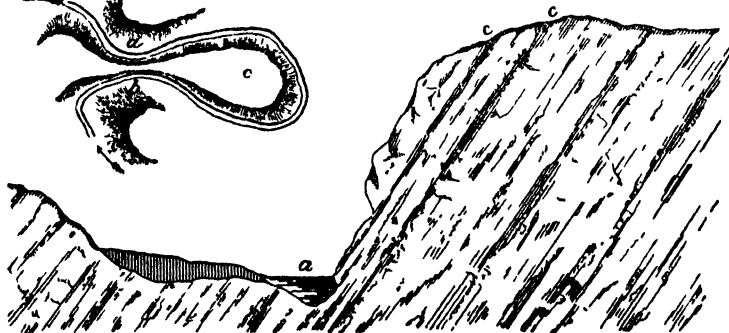


FIG. 49.



FIG. 50.



SECTIONS SHOWING EROSIVE ACTION OF RIVERS.

The stream having finally cut its bed, as indicated, continues to wear away the sides of the range at *a* and *b*, in the manner represented, until at last the isthmus is cut through, the channel is deepened, and, in the course of time, the old bed surrounding the peninsula is almost obliterated.

It is commonly remarked of the Victorian rivers which run through

basalt and Silurian claystone districts, that their beds are cut along the line of junction of the two formations. This is due to the softness of the argillaceous claystones. Rain falling on the claystone ranges would first settle in the hollows of the line of junction, decomposition and disintegration would follow, and a channel would soon be cut in the softer rocks in the line of the depression.

The diagrams just given are from actual sections by Brough Smyth, but they only represent the simplest changes met with. A deep study of and intimate acquaintance with the geological disturbances which have been developed in a district, must always prove of immense assistance to the seeker after deep leads. It is, in fact, the point which demands the greatest attention, and ought to form the basis for all mining operations undertaken in this class of deposits. The following diagrams, therefore, cannot fail to be found exceedingly interesting and instructive. They are due to Captain Couchman, chief mining surveyor in the Maryborough district.

In that district, the shallowness of the auriferous alluvions on the summits and slopes of the clay-slate ranges, and the regularity of their

FIG. 51



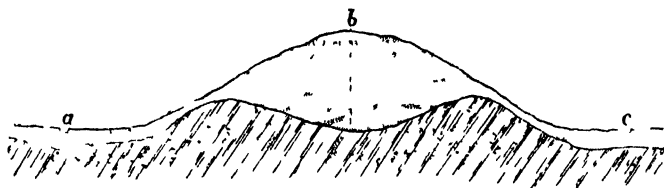
DEEPLY-SILTED GULLIES FAR FROM PRESENT STREAM.

direction, are generally remarkable. They afford also evidence of the great changes that have been developed in many of the drainage channels. They are commonly found in or near the present water-courses, but there are many exceptions to this rule, one of the most important of which is illustrated in Fig. 51. Here the old valley beds have been silted up to high levels, forming wide alluvial flats, and the wash-dirt *bc* is found far from the present water-course *a*, though still in the same flat. This feature seems to have originated in a constant shifting of the bed of the stream during the time that the valley was being silted up, as evidenced by the successive layers of clay, shingle, and sand.

It sometimes happens that after old channels containing auriferous alluvial deposits have been silted up in a great measure, deeper valleys are worn down on either side, so that what was formerly a valley is made to assume the appearance of a hill, as shown in Fig. 52. The hill from which this illustration is taken has the outward appearance of a

clay-slate spur, but really consists chiefly of alluvial deposits, in which gold has been found immediately under the summit; *a c* are modern water-courses, *b* is wash-dirt.

FIG. 52.

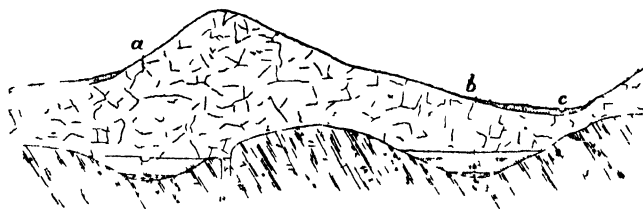


GUTTER HIDDEN BY ALLUVIAL HILL.

Where flows of basaltic lava have wholly or partially covered the Silurian claystones, &c., the courses of the modern streams often vary widely from the direction of the older valleys beneath, though exhibiting conformity in the general tendency of the main drainage between the watersheds and the sea or the chief rivers.

An instance of complete capping with basalt is shown in Fig. 53.

FIG. 53.



GUTTER COMPLETELY CAPPED BY BASALT.

New river channels *c* have cut their way in the basalt in the lines governed by the depressions and slopes on its surface, while beneath, as at *a* and *b* deep leads of auriferous alluvium occupy the beds of the ancient river system.

Where the basalt is of an indurated description, and its flow has been restricted within the limits of a narrow valley, the formation will assume the character illustrated in Fig. 54. The basalt was confined to the bed of an alluvial valley in the older sedimentary rocks, probably bounded on either side by hills sloping in the direction of the dotted lines, the volcanic rock conforming with the level of the horizontal broken line. The basalt has succumbed to the degrading influences of the elements much less readily than the adjacent claystones, and thus newer valley beds have been eroded at each side to a lower level, and the ancient river

channel with its coping of basalt is left standing as a hill or small plateau; *ac*, wash-dirt; *bd*, existing water-courses. In all parts of Victoria where the leads have been explored, it has been noticed that the existing main-drainage channels have been determined by the form of the surface of the Palæozoic rocks. Recent lava-flows have modified the

FIG. 54.

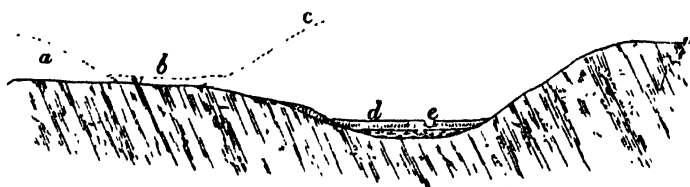


GUTTER BENEATH ERODED BASALT HILL.

valleys, and varied the courses of the streams; but the main features, resulting from the aqueous agencies at work previous to and during the formation of the leads, remain, though difficult to trace.

With regard to the shallow auriferous hill deposits before spoken of, miners commonly suppose them to be of more recent formation than the leads in the valleys, and consider them to result from a later disintegration of richly auriferous quartz veins in the immediate vicinity. This may possibly be a correct view of some of those deposits which do not lie directly on the bed-rock; but where they are in actual contact with the Silurian strata, and the gold and quartz detritus bear a water-worn appearance, attesting a process of friction and disturbance such as the materials forming the valley leads have been subject to, there is no room for such a theory, and they must rather be taken as special illustrations of the extraordinary changes that have been effected in the river-beds of the older sedimentary rocks by ordinary agencies still at work. The chief causes which have governed the changes that have

FIG. 55.



SECTION SHOWING CHANGE OF RIVER-BED.

taken place in many of the valleys in schistose formations may be briefly stated as the greater disintegration of the argillaceous rocks over the others in the same drainage area; the tendency of rocks to wear chiefly in the direction of the dip of the strata (as illustrated by Fig. 55), owing

to the more direct and powerful influence of the elements upon the exposed edges; the more swift erosion of the steepest hills, on account of the rapidity of the drainage and the redoubled scouring action of rain-storms on the slopes; the shifting of water-courses from one valley to another lower, by a channel being cut through the dividing range; and the action of streams from tributary valleys in a lateral direction across the main valleys. In Fig. 55, *ac* are old hills; *b*, an old valley; *d*, a new valley; *e*, existing water-course.

The positions occupied by the gold in the leads are very diversified: in some strata, it is scattered throughout from surface to bed-rock; in others, it is peculiar to the thin layer of quartz pebbles and alluvial drift lying on the bed-rock; in many places, false bottoms occur.

It might be supposed that in the case of a rich stratum of gold being found actually lying upon the bed-rock, the whole of the auriferous deposit had accumulated at one time, and that the barren alluvium was subsequently superimposed. This was, however, by no means the case. It is evident that the flow of water and earthy matters contained from first to last a certain proportion of atoms of gold, whose size was governed by the dimensions of its original matrix, and the degree of abrasion and reduction to which it had been subjected, the latter depending on the force of the motive power, and the obstacles met with. Having stopped in some hollow, the superior weight of the precious atoms would cause them to sink through the moist surrounding matters till a hard layer was met with. The recurrence of this process would constantly add to the deposit, the gold always gravitating towards the bottom, quickly or slowly, according to circumstances. If water was held long enough to soften the previously deposited material, the bottom would soon be reached; but if the water passed off at once, the gold would remain principally on the surface of the last layer, or partly scattered through the newly deposited mass, and thus the heaviest and smallest particles—their shape being favourable—would continue to descend, so that in course of time the largest proportion must accumulate in the lowest parts, and above strata that are impermeable to water,—generally the bed-rock. Time and circumstances may render it probable that several strata of gold may be found, independent of a considerable amount pervading the whole alluvial deposit, though not having had an opportunity of settling down.

Surprise has often been expressed at the discovery of large water-worn boulders of quartz and other rocks in the alluvions of little channels, where the stream is insignificant even in wet weather, and certainly incapable of moving them under ordinary circumstances. Their abrasion and transportation are generally to be attributed to floods, often occurring where the formations resemble those shown in Figs. 48, 49.

Sometimes also the detritus has been derived from drainage areas no longer in connection with the modern stream. When a lead is struck, much uncertainty must surround its relation to other leads, until it has been opened up to some extent. Short of actual exploration, perhaps the safest guides are the character of the detrital matter in the gutter or bed of the lead, and the direction of the fall of the water ; but they are not infallible, for the colour, character, and even mode of occurrence of the strata may vary greatly in the same lead, owing to the influence of tributary streams, and water may in a similar manner be introduced in large quantities, and thus give rise to serious errors. It is very necessary to bear in mind that the gutter or actual bed of the stream does not represent by any means the distribution of the auriferous ground ; the width of the lead is not confined to the gutter, and sometimes the yield of gold is greater from other parts of the lead than from the gutter itself.

The following remarks on the auriferous gravels of the Sierra Nevada are condensed from the elaborate and exhaustive treatise on the subject by Professor J. D. Whitney.

The whole series of phenomena observed in the Sierra Nevada gives ample grounds for the belief that the metamorphic rocks of the range,—the bed-rock in fact,—were the original home of the gold now so generally distributed through the gravel. Obviously the mode of occurrence of the auriferous gravels themselves shows clearly that their presence is entirely dependent on that of a body of slaty rocks, enclosing great numbers of quartz veins, which in many cases are sufficiently well supplied with gold to repay the expense of mining. In short, the quartz mining districts and the gravel region are essentially one and the same tract of country, as will be apparent on inspection of the various maps accompanying Whitney's splendid volume.

Some very fine gold has been carried, no doubt, to a considerable distance from its native bed, and deposited upon strata which were not in the least auriferous ; but the coarser portion of the gold remains either on or in close proximity to the place where it originated.

That the distribution of the gold in the gravel should be irregular, and that it should be widespread, are conditions which seem quite in harmony with its origin. Quartz veins are very generally distributed through the bed-rock series ; a large portion of these contain some gold, many of them are rich enough—in places, at least—to pay for working. But it appears to Professor Whitney very clear that the older gravels have been formed from a material considerably richer in gold on the average than is the present bed-rock of the region.

As to the question whence came the gold contained in the quartz veins, Whitney would derive it from the enclosing and surrounding rocks

themselves. There are abundant facts to prove beyond all doubt that gold is distributed in minute quantities far and wide throughout the great mass of these metamorphic rocks, entirely independent of all the quartz veins. Indeed, it may well be questioned whether there is so much as a single ton of metamorphic rock of the auriferous series above the level of the sea in the Sierra Nevada in which a sufficiently delicate chemical analysis might not detect a trace of gold; and it can hardly admit of a doubt that the aggregate quantity of the gold which is thus distributed through the mass of the rock is incomparably greater than that which exists in the quartz itself. There is thus no lack of resource from which the gold in the quartz may have easily been derived; while, on the other hand, the general character of the quartz veins themselves appears to him to furnish strong evidence in favour of the view that the gold in them has been thus derived. They are not often what are technically called "fissure veins." They are generally intercalated between the planes of stratification near the slates, and irregular bunches and lenticular masses of limited extent are common among them. Furthermore, the rocks in many localities are penetrated in every direction by little irregular quartz veinlets and stringers, which often carry gold, and are sometimes in spots extremely rich, even when less than an inch in thickness.

Throughout the gravel region, the evidence constantly presented is cumulative and overwhelming that these gravel banks are the work of streams of fresh and running water. But then arise the great questions, Whence came these streams, and where did they go? In answer thereto, the theory which first gained extensive credence was that of the former existence of a single great river, flowing south-easterly for hundreds of miles along what is now the south-western slope of the Sierra, and nearly parallel with the axis of the range, before the mountains were uplifted. And the supposed channel of this great stream, now filled with gravel, was called the "blue lead."

This theory was founded, of course, upon the supposed existence of a narrow belt of deep gravel banks, stretching across the country in a south-easterly direction. But later, when it came to be more generally understood that the deep-gravel banks were not all confined within so narrow a belt as this theory would require, the theory was modified far enough to admit the existence in the gravel period of two or three large rivers with their branches, the courses of the main streams, however, being still supposed to be nearly parallel, and the general direction of their flow to be south-easterly.

The first objection to this blue-lead theory lies in the fact that from the Middle Yuba to the Mokelumne River, including the best-known and most important gravel-mining region in the State, there is no well-

defined belt or belts of deep gravel banks stretching N.W. and S.E., such as must necessarily underlie the very foundation of this theory.

The single fact of the non-existence of any such definite north-west and south-east belt or belts of deep gravel banks as have been supposed, though perhaps insufficient of itself to positively disprove the blue-lead theory, does nevertheless completely remove the very foundation on which it was built.

But there are other and more positive objections to it. Incidentally, Whitney mentions another important fact connected with this theory. The term "blue lead" derived its origin, in part at least, from the blue gravel, and there has always been an intimate association in ideas between the two, many even going so far as to believe that the blue gravel was characteristic everywhere of the blue lead, and was not found outside of it. But the fact is that the blue gravel is scattered here and there all over the gravel region, though its quantity is ordinarily the greatest where the banks are heaviest. It cannot be said to be characteristic of any particular channel or set of channels. And Whitney is thoroughly satisfied that the peculiar dark-bluish colour which gave rise to the name "blue gravel" is due, in the vast majority of cases at least, not to any original peculiarity in the character of the rocks from which the gravel was derived, nor to any peculiarity in special streams which once flowed in particular channels, but simply to local chemical action in the mass of the gravel itself subsequent to its deposition.

The south-western slope of the Sierra Nevada, at the commencement of the gravel period, was, in Whitney's opinion, a broad, gently undulating, and moderately hilly country, having in all probability exactly the same gentle average slope which exists to-day south-westerly towards the valley,—a slope which for great distances ranges from 100 to 130 ft. to the mile,—and having at the same time its greater drainage basins, already marked out where they exist to-day, by the occasional ridges and spurs which traversed, sometimes for considerable distances, the rolling hilly country, though rarely rising so much as 1000 ft. above the general plane. High up towards the summit of the range, indeed, the country grew gradually rougher; and here was probably more or less of cañon and of precipice, though it is not at all likely that there was anything comparable in magnitude with many of the modern cañons.

There may possibly, indeed, have been among the peaks occasional gorges 2000 or 3000 ft. in depth: but it seems hardly possible that any such gorges then could have been continuous for many miles, and they were the results, in all probability, not so much of denudation or erosion, as of the original upheaving forces which built the range and raised the lofty peaks.

Among these loftier summits, the streams which slowly gathered the

ancient auriferous gravel took their rise. Thence they flowed south-westerly across the smoother sloping country, the larger streams draining respectively nearly the same regions as now, and in the same directions, and finding their way by the shortest practicable routes to the valley, which, however, may then have been a lake or sea.

This appears to Whitney to be the general outline of the manner and means by which the ancient auriferous gravel was slowly accumulated, its materials—i. e. its gold, as well as its boulders, pebbles, and sand—being derived from the bed-rock over which the streams ran, and from the quartz, &c., which it contained. The longer such a process continued, the more the streams would wander, the greater would be the quantity of gravel accumulated, and the greater the actual area over which it would be spread. He thinks this process continued until the advent of the volcanic era, when the enormous quantities of volcanic matter which were spread far and wide over the western slope of the Sierra, buried beneath their mass the previously accumulated gravel, as well as the rocks from which it was derived, and were the chief and direct cause of the final ending of that gravel period.

Sections.—Sections of the strata cut through in reaching deep leads are very interesting, both to the miner and the geologist. Those hereafter given are principally on the authority of Brough Smyth, and refer to Australia :—

	ft. in.		ft. in.
1. Black soil	4 0	4. Surface soil	2 0
Red gravel	2 0	Stiff yellow clay	8 0
Red clay	13 0	Drift (dry)	13 0
Drift	34 0	Stiff yellow clay	9 0
Boulders	3 0	Fine sand	2 0
Cement and sand	3 0	Ditto, darker colour, with a little	
Auriferous drift	2 0	gravel	4 0
Bottom : Micaceous sandstone.		Fine white sand	4 0
	61 0	Cement of drift granite and	
		sand	14 0
2. Red clay	7 0	Arenaceous clay	23 0
Basalt	90 0	Fine yellow clay	3 0
Boulders	9 0	Coarse drift	12 0
Auriferous drift	3 0	Fine drift sand and clay	1 0
Bottom : Green slate.		White and yellow clay	2 0
	109 0	Fine drift (no water)	4 0
		Very tenacious white pipe-clay	3 0
3. Soil	7 0	Dark-yellow clay and grit	9 0
Red clay	8 0	Very fine running drift (much	
Black clay	35 0	water)	15 0
Decomposed basalt	10 0	Stiff blue and red clay	2 0
Black clay	11 0	Light coloured drift	1 0
Boulders	15 0	Stiff blue clay	1 0
Auriferous drift	8 0	Tough black clay mixed with	
Bottom : Schistose.		charcoal	5 0
	94 0	Blue clay	2 0
		Very tough clay	4 0

Red clay with streaks of white and yellow	112	0	Red clay with streaks of white and yellow	120	0	
Cement (false bottom)	2	6	Red sand (cement)	0	6	
Hard cemented granite drift ..	3	0	Coarse granite drift, very hard, with very little water	2	6	
Coarse gravel	3	0	Auriferous drift, colour of gold obtained	2	0	
Auriferous coarse drift gravel ..	7	0	Bottom : Yellow slate rock.			
Bottom : Soft white slate and pipe-clay.	270	6		261	0	
5. Surface soil	2	0	7. Surface soil	2	0	
Sandy clay	18	0	Clay and sand	38	0	
Brown sandy clay	18	0	Drift	14	0	
White sand	2	0	Clay	27	0	
Yellow clay	20	0	Drift	6	0	
White pipe-clay	2	0	Clay	11	0	
Red sandy clay	2	0	Clay, sand, and drift	15	0	
Dark-brown sandy clay	14	0	Clay mixed with drift	25	0	
Water	2	0	Black stiff clay	4	0	
Variegated sandy clay	9	0	Yellow and blue clay	18	0	
Red sandy clay	2	0	Red clay with streaks of white and yellow	102	0	
Yellow clay	10	0	Auriferous drift gravel mixed with cement	4	0	
Yellow sandy clay (more water)	19	0	Bottom : Soft white slate and pipe-clay.	266	0	
Red sandy drift	10	0	8. Surface soil	1	0	
Yellow clay	10	0	Red sandy gravel mixed with fine quartz	40	0	
Red clay with streaks of white and yellow	81	0	Fine quartz gravel, not water-worn	10	0	
Bottom : Yellow slate rock.			Very fine white indurated clay, with a slight appearance of stratification and verging towards cement at the base ..	43	0	
No drift, no gravel, and no gold.	221	0	Compact sand	3	0	
6. Surface soil	2	0	Quartzose gravel 3 to	5	0	
Stiff yellow clay	18	0	Auriferous drift resting on Silurian bed-rock	5	0	
Coarse granite drift, very hard	4	0		107	0	
Stiff yellow clay	16	0	9. Loam and clay 2 ft. to	4	0	
Very fine sand	4	0	Ferruginous cement .. 2 ft. to	4	0	
Coarse granite drift with clay and fine sand	14	0	Gravel, sand, and angular fragments of quartz .. 20 ft. to	30	0	
Stiff variegated clay	2	0	Fine compact gravel	2	0	
Hard red granite drift	4	0	Cement (sometimes wanting)			
Very stiff variegated clay ..	3	0		3 in. to	1	3
Hard red granite drift	4	0	Auriferous drift 5 ft. to	12	0	
Red and white clay, very hard	4	0	Very fine compact indurated clay (false bottom) 20 ft. to	50	0	
Stiff yellow clay	9	0	Auriferous ferruginous cement lying on clay-slate (true bottom) 6 ft. to	10	0	
Hard granite mixed with a little clay	7	0	Maximum	113	3	
Hard red granite drift	1	0				
White clay and fine sand ..	2	0				
Fine soft yellow clay	2	0				
Granite drift and clay	3	0				
Soft white and yellow clay ..	2	0				
Granite drift with a little water	9	0				
Red and yellow clay	15	0				
Red, white, and yellow clay ..	2	0				
Very fine (dry) light-coloured clay	3	0				
Fine dark-coloured drift with charcoal	3	0				
White clay with quartz	3	0				

Only the lower part of either stratum in No. 9 is very rich in gold. Towards the sides of the gutter, near the bounding ranges, the two bottoms do not seem to be found; but this is not made clear. Invariably the two bottoms are met with wherever a tributary enters the main lead. Probably the two gutters do not follow the same direction, or the lower lead may be wider than the upper.

	ft. in.		ft. in.
10. Black surface soil	0 9	11. Black surface soil	2 0
Clay with quartz-gravel	10 0	Clay with quartz	7 0
Basalt	48 0	Basalt	52 0
Stiff clay	8 0	Stiff black clay	9 0
Quartz drift	2 0	Quartz drift	3 0
Wash-dirt (quartz-gravel) 9 in. to	2 0	Wash-dirt 8 in. to	3 0
	70 9		76 0

Fall of the gutter: 3 ft. in every
100 ft. Feruginous cement
sometimes met with.

Width of gutter: 200 ft. and not
fully explored. Fall: 3 ft. in
every 100 ft. Ground very wet.

12. In the Empress claim, the wash is composed of quartz pebbles in various stages of attrition, sandstone pebbles highly water-worn, and large masses of jasperoid quartz, evidently derived from a neighbouring reef that has been intersected by the tail-race of the claim. The following is a section of the deposits exposed in the open cut:—

	Ft.
Surface soil, gravel, and sand, in places thicker, and containing blocks of basalt, derived from the ridge at the back	18
Clay	3
Clay containing stems of small trees	3½
Carbonaceous clay, containing leaves, &c.	4½
Clay	2½
"Wash" impregnated with oxide of iron	18
Clay	3½
Wash	30
Total thickness of deposit	<u>83</u>

The bed-rock is a Silurian slate, yellow in colour, and much decomposed. The stems of small trees contained in the second layer of clay are all more or less flattened, and in various stages of decomposition, at times the wood being almost of its natural colour, while occasionally specimens occur converted into lignite, and exhibiting a fracture like charcoal. Many of the wood specimens show joints. The most abundant leaf found in the carbonaceous clay is apparently a variety of *Pteris esculenta*, which now grows abundantly in the locality. In places, this leaf seems to form the bulk of the organic matter which is irregularly distributed through the 4½-ft. clay band.

13. The drift of New Chum Hill is composed of quartz pebbles of various degrees of roundness, also very water-worn sandstone pebbles, sand, clay, and included fragments of the slate bed-rock. The following section is shown in the Emperor claim :—

	Ft.
Basalt	1½
Clay	8
Carbonaceous clay, leaves, fern-stems, &c.	8
Pipeclay	8
Sand	4
Clay (very fine, like fire clay)	1½
Clay	5
Carbonaceous clay	35
Clay (very white and fine)	15
Wash-dirt	30
Total thickness of deposits	<u>116</u>

14. The section of the centre portion of the Nine Mile Creek diggings is given by Gipps, as follows :—

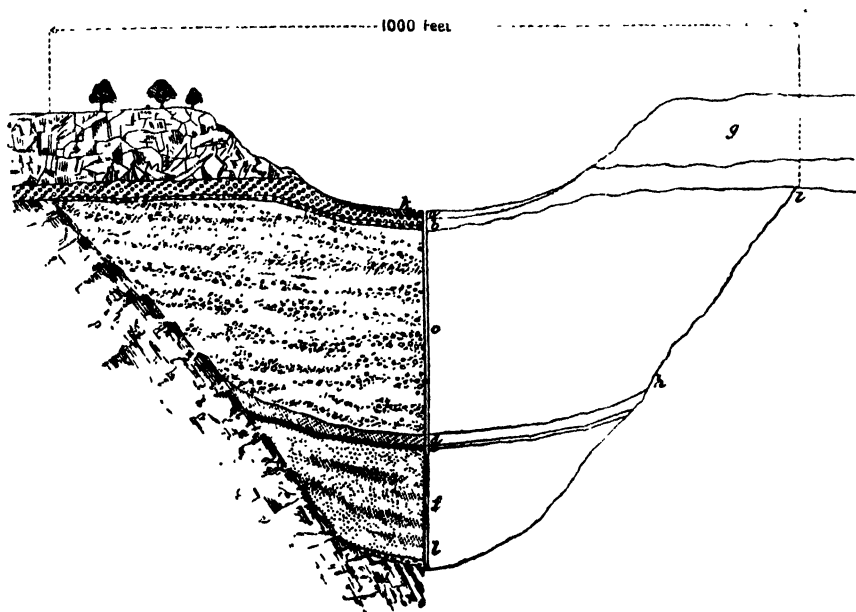
	Ft.
Dark-brown surface soil, mixed with floating boulders of basalt, and large blocks of concretionary quartz cement, containing gold	7
White and grey clay bands	20
Brown-coloured lignite	18
Sand	3
Fine drift, composed of small quartz and slate pebbles, seamed with thin layers of iron-stone cement, containing gold	65
Coarse drift, composed of big boulders of quartz, jasper, and iron-stone (denoting a swift large stream on the bottom of the original valley), containing gold	6
	<u>119</u>

At base, is a soft rotten Silurian slate, seamed with quartz and iron-stone. This measurement gives a total depth of 119 ft., of which 78 ft. is auriferous.

Fig. 56 represents a section of the leads at Ti-tree Creek, Moorabool, which are believed to be of Miocene age. All the ordinary gold-workings, as here shown, are carried on in a recent drift, immediately underlying the basalt and overlying the older strata. The gold is fine, but generally distributed, and has been profitably worked for several years. The reference letters indicate as follows :—*a*, black clay and loam ; *b*, gravel containing considerable quantities of fine gold ; *c*, 320 ft. of cement, quartz boulders, &c. ; *d*, 9 ft. of drift ; *e*, dark clay with imbedded trees and fine gold ; *f*, cement drift with layers interspersed with gold ; *g*, basalt ; *h*, schist ; *i*, reef containing coarse heavy gold ; *k*, creek ; *l*, shaft 460 ft. deep. The Golden-Rivers Company sank 460 ft. in order to reach the bottom, in the hopes of finding richer ground than the lead on the false bottom which they were working. After passing through a

drift containing small well-rounded pieces of quartz and a great deal of pyrites, a seam of black clay was struck, enclosing fossil trees and a little fine gold, and beneath this a thick layer of grey sandy clay with small fragments of fossil wood. On the hard yellow sandstone bed-rock, every

FIG. 56.



MIOCENE LEADS AT TI-TREE-CREEK.

sample taken from the lead yielded gold, and the precious metal was found to be present wherever the true bottom was laid bare, but not in sufficient quantity to pay for working.

Another section near here shows :—

Upper basalt rock, about 25 to 30 ft.

Pliocene gravel, about 50 to 60 ft.

Miocene gravel, &c. (the "false bottom"), gravel, sand, clay and boulders, with fossil leaves and wood, about 400 ft.;

Silurian slates, &c.

A third section runs :—

Upper basalt, 49 ft.

Sandy Pliocene grit, 10 to 15 ft.

Upper coralline limestone, 13 ft.

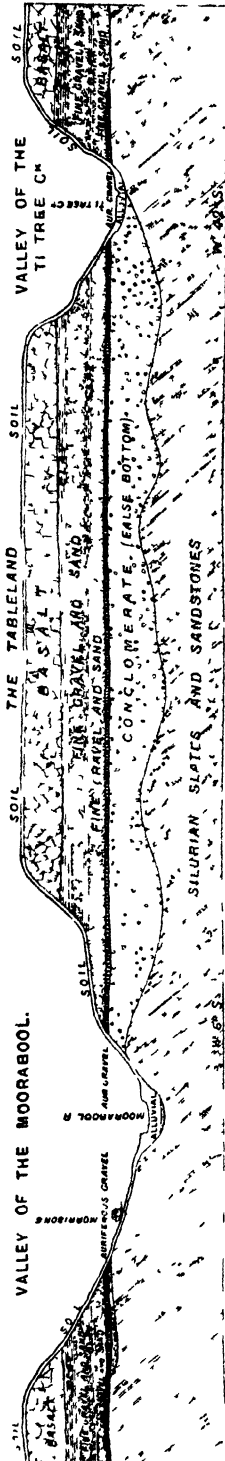
Older basalt containing bands of hard compact limestone with fossils. } Miocene.

Sandy limestones, with fossils, 30 ft.

Rounded quartz pebble drift, and hard siliceous conglomerate rock with fossil wood, lower part gravel and boulder drift, 90 ft.

Silurian slate and sandstone with quartz veins.

FIG. 57



SKETCH CROSS-SECTION OF LEAD AT MORRISON'S.

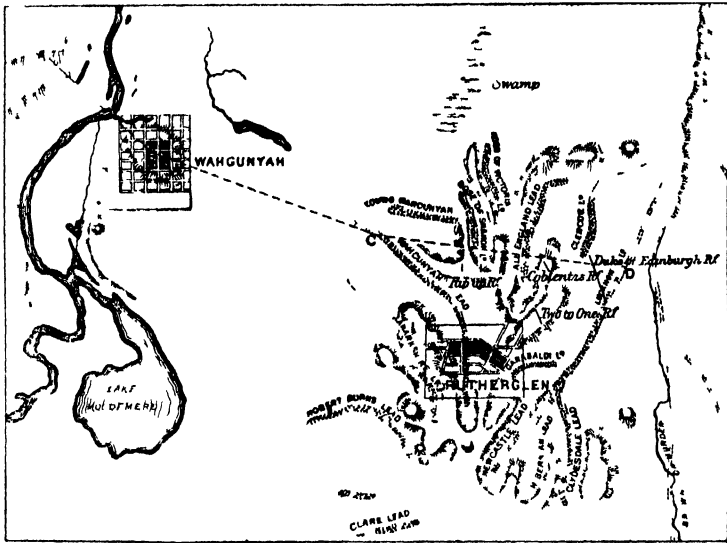
SCALE horizontal, 8 chains = 1 in. ; vertical, 100 ft. = 1 in.

Fig. 57 illustrates a cross section of the lead at Morrison's, the scale being : horizontal, 8 chains = 1 in. ; vertical, 100 ft. = 1 in.

Figs. 58 and 59 show a plan and section of the leads in the neighbourhood of the Murray river, Victoria, copied from Brough Smyth's work. The horizontal scale is 30 chains to 1 in. ; the vertical, 200 ft. to 1 in. The break between the two portions of Fig. 59, representing the area occupied by the probable ancient bed of the Murray river, may be understood to have a width of 150 chains. The letters of indication on Fig. 59 have the following reference :— *a*, red-chocolate coloured soil, with a few rounded and sub-angular quartz pebbles ; *b*, the surface of this flat consists of a tenacious marly clay, overlying the ordinary river drifts, probably of no great depth ; *c*, banks consist of nearly perpendicular rock about 50 ft. above the present level of the stream, composed of brown, white, and red sandstone and slate-rock ; ferruginous quartz veins particularly numerous, having every direction from perpendicular to horizontal ; *d*, red soil, overlying clay-slate and sandstone rock ; *e*, marly clay, used in brick-making ; *f*, surface covered with vast quantities of rounded and sub-angular quartz pebbles ; gold has been found in the stone ; *g*, a drain in this gully shows a considerable quantity of surface limestone nodules ; *h*, quartz pebbles : brown and red clay soil ; *i*, total depth of shaft said to be 235 ft. ; the width of the lead never exceeds 40 ft. ; when it runs out in a fine drift 2 ft. 6 in. thick, the wash-dirt is about 5 ft. 6 in. thick ; bed-rock of the usual schistose formation ; *j*, a quartz-prospecting shaft

sunk on this hill shows 3 ft. of alluvium overlying the sandstone and slate rock, intersected by several very narrow quartz veins, running down quite perpendicularly ; *k*, quartz pebbles very plentiful on this hill ; no exposure of rock ; *l*, shaft about 60 ft deep ; the lead very poor and unimportant ; *m*, the veins are very small, the largest not more than 4 or 5 in.

FIG 58.



PLAN OF MURRAY RIVER LEADS

thick, *u*, this has been one of the most productive leads in the district; the depth of the last shaft sunk was 285 ft., the wash-dirt was 2½ to 5½ ft. thick, and the width of the lead was 40 to 80 ft.; *o*, there is no exposure of rock on the sectional line, but on the apex of the hill several quartz veins have been extensively worked; *p*, this lead seems to have originated in the quartz veins intersecting the apex of the low Silurian range; the depth of sinking is 260 ft., with wash-dirt 60 ft wide and 2 to 5 ft. deep; *q*, close here is a thin deposit of alluvium covering schistose rock intersected by quartz veins; *r*, a highly payable lead for a distance of 154 ft. west from the shaft; drift chiefly composed of sub-angular fragments of quartz.

In the main lead, at Bendigo, are two layers of cement, one formed on the top of the wash-dirt, 1 to 7 ft. from the bed-rock, where the gold is so fine as to be called "paint-gold," yet so abundantly and thoroughly disseminated through the rock that it averages 5 to 6 dwt. per ton, and sometimes reaches $1\frac{1}{2}$ oz. It is exceedingly hard, and has to be broken down with sledge-hammers, as the loose, sandy character of the over-

FROM THE TOWN OF



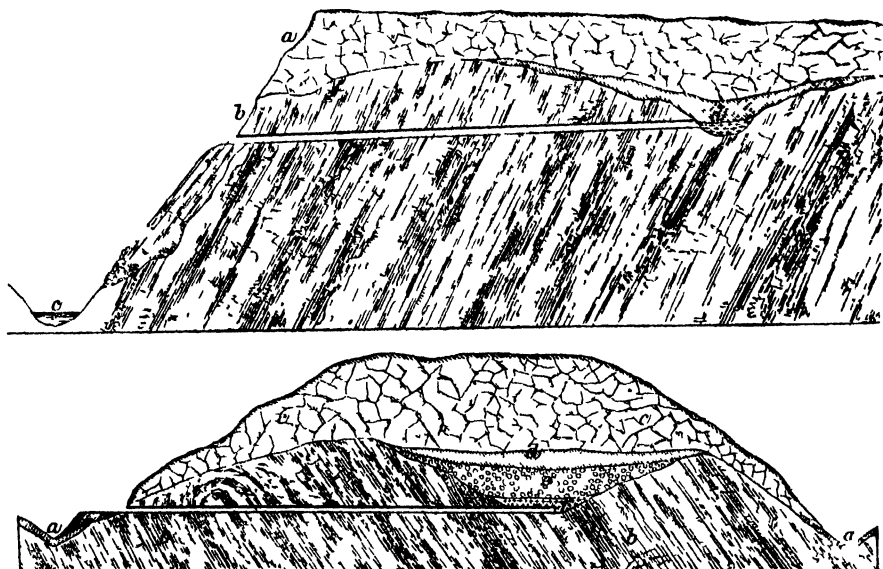
SECTION OF MURRAY RIVER LEADS.

lying stratum precludes the use of gunpowder. The other layer of cement lies on the bed-rock, and is only 1 to 4 in. thick. Some of it is brittle and even friable, and it has been found exceedingly rich in places, seams of it running for 18 in. into the bed-rock, while its average yield has been 1 oz. per ton. The strata are about as follows :—

	ft.	in.
Sandy loam	1	6
Sandy 10 ft. to	12	0
Gravel like road-metal	30	0
Sand in layers	30	0
Tough clay 4 ft. to	6	0
Fine drift sand	11	6
Gravel and boulders 3 ft. to	9	0

In the Daylesford district, the lines of the outflows of volcanic matter can be distinctly traced, notwithstanding the changes effected by denudation. The alterations in the contour due to riverine action are partly referable to the great extent of heavily wooded country lying to the south, on and near the Main Spur, where rain falls nearly all the year

FIGS. 60 AND 61.

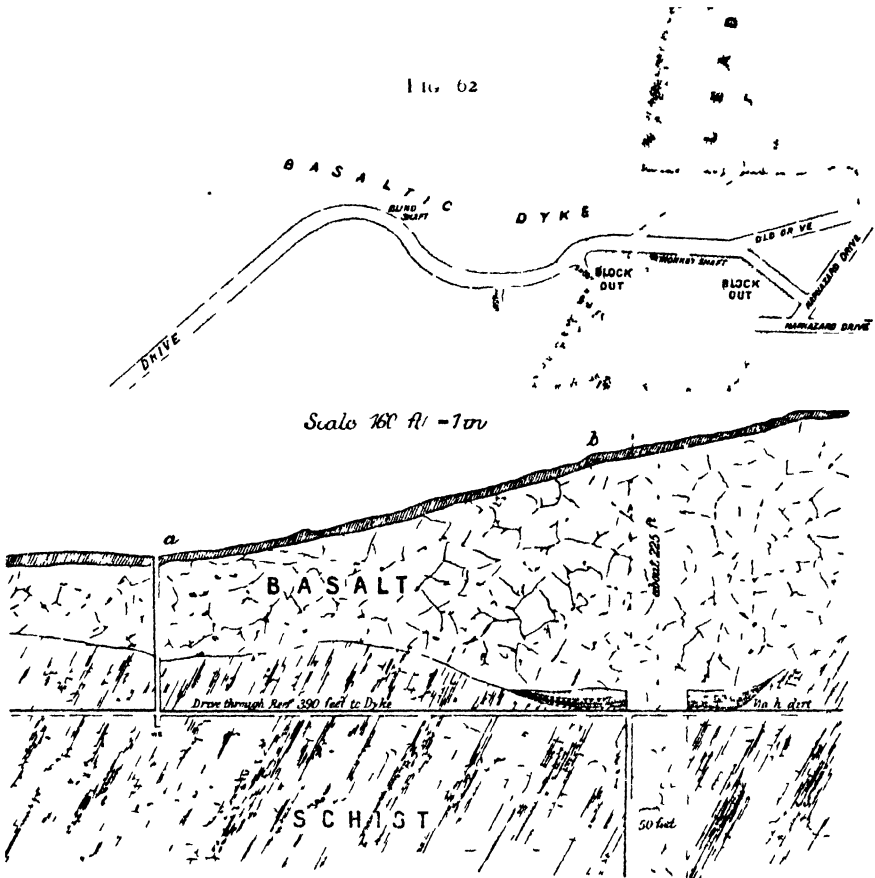


DEEP LEADS IN THE DAYLESFORD DISTRICT.

round. The streams fed from this source have cut deeply into the basalts and Palæozoic clay-stones, and the leads or ancient river-beds have to be sought for in the bounding ranges of the present valleys, rather than deep beneath the surface of the latter. The modes of occurrence of most of the deep leads in this district are represented by the actual sections depicted in Figs. 60 and 61. In Fig. 60, *a* is basalt,

b, Palæozoic claystones, &c.; *c*, modern water-course; *d*, lead. In Fig. 61, *a*, are modern water-courses; *b*, Palæozoic clay-stones; *c*, basalt; *d*, clay and sand; *e*, sand and quartz pebbles; *f*, auriferous drift.

One property here, shown in Fig. 62, possessed a novel feature, in the shape of a dyke or pipe of basalt, which puzzled the miners considerably. Believing it to be an ordinary cliff, they sank down beside it for about 140 ft. at *a*, but found no bottom, and they afterwards drove through it, and discovered the bottom on the other side. The dyke is



BASALTIC PIPE CUTTING LEAD.

about 50 ft. thick, and the overlying basalt has a considerable thickness. It is possible that this dyke or pipe may be the outlet of an ancient crater. From a report, it seems that the shaft No. 1 was sunk to a depth of 130 ft., passing through surface soil 10 ft., basalt 80 ft., and Silurian slate 40 ft. A drive was then opened out and pushed 390 ft. due east till reaching the basaltic dyke running S.E. After sinking 170 ft down the

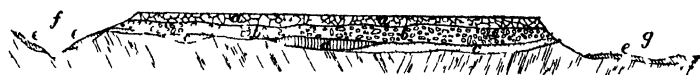
western edge at *b*, and finding no termination of the intruded rock, the original drive was continued S.E., and the gutter found abruptly ending at the edge of the dyke.

In the lower part of the Coliban, it is noticed that wherever a basalt capping exists, the drift beneath is more or less auriferous. A shaft in that district shows,—Surface soil, 1 ft. ; clay with fragments of quartz, 6 ft. ; quartz-gravel, 13 ft. The bottom is pipe-clay ; width of gutter 6 to 8 ft. ; thickness of wash-dirt, 1 to 3 ft. ; fall of gutter, 3 ft. in every 100 ft. The wash-dirt consists of quartz boulders, clay and sand, and a portion of the pipe-clay bottom. The gold is coarse and not much water-worn.

A neighbouring shaft has cut through,—Black soil, 6 ft. ; basalt, 80 ft. ; stiff clay, 68 ft. ; drift, 6 ft. ; coarse pebbly wash, 3 ft. Width of gutter about 20 ft. ; the gold not much water-worn.

At Vaughan, near the junction of Fryer's Creek and the river Loddon, the auriferous leads occur as shown in Fig. 63. The reference letters

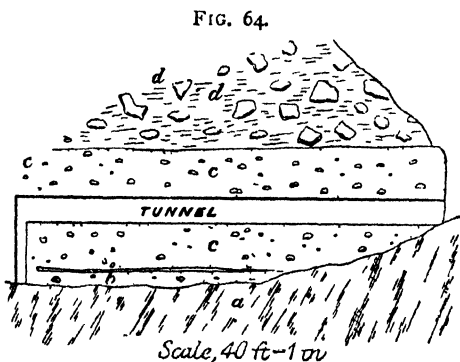
FIG. 63.



LEADS ON FRYER'S CREEK.

have the following significations :—*a*, basalt ; *b*, gravel with large quartz boulders ; *c*, auriferous drift ; *d*, cement ; *e*, alluvial ; *f*, river Loddon, *g*, Kangaroo Flat.

In Fig. 64, is seen a diagram of a tunnel driven through auriferous strata on the Upper Dargo river. The "rim-rock," or edge of the ancient channel, generally to be observed in the terrace deposits occurring at various heights above the present river beds, has been cut through to a depth of several feet. The reference letters are—*a*, Silurian ; *b*, plant-bed ; *c*, auriferous quartz-gravel ; *d*, basalt talus. The total thickness of the gravels is about 30 to 40 ft. They consist principally of water-worn vein-quartz and pale sandstone, with a few pebbles of quartz.



LEAD ON UPPER DARGO RIVER.

The gold is generally fine, and was disseminated throughout the gravels traversed by the tunnel in sufficient abundance to pay the working

expenses of prospecting. The specimens found in the plant-bed have been determined as of Miocene age. The careful examination and study of these Miocene rivers becomes of immense importance, now that they are proved to be auriferous.

Some information concerning the auriferous Tertiary beds at Newtown Hill is to be gathered from the shafts undertaken recently in that district. The first was put down through drift and ferruginous cement, 20 ft.; bottomed on blue slate reef. No. 2 sank 67 ft. through drift and large rounded quartz boulders, bottomed on slate reef, sank in slate to 82 ft., drove in it eastwards 90 ft., jumped up 15 ft., and broke into a wash yielding fine and (so-called) "coarse" gold, the latter but little water-worn. No. 3 shaft was sunk through strata similar to No. 2, bottomed at 85 ft. on blue slate reef, opened out at 94 ft., drove eastwards 72 ft., descended 5 ft. on to slate reef at 120 ft., sank 12 ft. on to slate, and at 180 ft. sank 31 ft. on to hard slate reef. The "reef," standing on edge, had an easterly slope throughout and a thin capping of auriferous drift. Other 3 shafts were sunk on the southern slope of Mercer Hill. No. 1 was through surface soil 10 ft., brown burnt clay 40 ft., drift and boulders 15 ft.; bottomed at 65 ft. on blue slate reef sloping east. No. 2 penetrated surface soil 4 ft., burnt clay 50 ft., drift with basaltic boulders 37 ft., fine gravel drift 6 in., blue slate reef 21 ft.; opened out at 112 ft. 6 in.; drove east 65 ft. and broke into wash, the reef still inclining east 1 to 3; drove another 85 ft. in boulder drift, leaving the wash under foot. No. 3 section is as follows:—

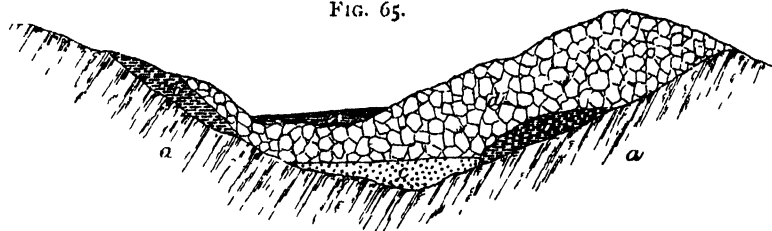
										ft
Alluvial	Surface soil	..	..	..	..	..	..	..	4	
Newer Pliocene ..	Brown sandy clay with charred wood	..	..	..	..	..	..	..	45	
Newer Volcanic ..	Vesicular basalt	..	..	..	..	..	..	..	12	
	Rounded quartz-pebble drift, with a shingle of quartz, gneiss, elvanite, and bluish-black fissile metamorphic slate, occasionally streaked with quartz								7	
Older Pliocene .. }	Drift of small rounded grains of transparent quartz, in a base of white felspathic clay								30	
	Coarse quartz-gravel								2	
	Whitish sandy clay								6	
	Sand drift with quartz-gravel								18	
	Sandy clay								2	
										126

The material composing the older drift has undoubtedly been derived from granite and the contiguous metamorphic rocks, and the lower beds of the last section are very similar to those of the Great Western lead, where the auriferous leads rest on granite. The "coarse" gold mentioned above should probably be termed "scaly" rather. The presence of coarse gold is supposed to imply the proximity of the Older Palæozoic

rocks, while fine gold may be carried a considerable distance from its source, and occur in small quantities overlying indiscriminately rocks of any formation older than the drift with which it is associated. If it can be shown that the bed-rock at Newtown Hill is a carbonaceous rock, with or without overlying Miocene beds, it is deemed useless to expect remunerative results from gold searching; but if Silurian rocks underlie the Tertiaries and Newer Volcanic beds, then, say geologists, "the principal conditions of a gold-field may be admitted to be present."

From the Stony Creek down to and through the Glenmaggie preemptive right, are distinct indications of an old river-bed now filled up with basalt. In the natural sections exposed, the basalt rests occasionally

FIG. 65.



RIVER-BED FILLED BY BASALT.

directly on Silurian formation, but in other places is separated from it by thick beds of intensely hard, cherty, silicious rock and conglomerate. The silicious rock thins out as the Silurian rocks rise on either side of the channel, and does not appear to occupy the river-bed, which probably contains a gravel deposit, as a few indications of such are met with. This gravel would therefore seem to be somewhat more recent than the silicious rock, and the section, if exposed, is believed to be as shown in Fig. 65: *a*, Silurian; *b*, silicious rock; *c*, gravels in the lead; *d*, basalt; *e*, most recent alluvial deposits. The actual sections are shown in Figs. 66, 67, and 68:—*a*, Silurian; *b*, silicious rock; *b'*, gravel of apparently the same age; *c*, gravel in lead; *d*, basalt; *e*, alluvial deposits. Rolled boulders of the silicious rock, polished like glass, are frequent in some places. There is also a deposit of gravel, forming hills above the level of the volcanic formation, but traceable to lower elevations and beneath the volcanic rock. This gravel may be of the same age as the silicious rock beds, or the gravel in the old channel, but as yet no definite opinion can be expressed. The modes of occurrence and doubtful character of the relations of the gravel and silicious rock are illustrated in Fig. 67. The volcanic rock, when solid, is a dense, hard, dark basalt, but is usually much decomposed. To the south of the Glenmaggie Creek, are tracts of country consisting of quartz-gravel and ferruginous cement, somewhat

higher than the basaltic hills, and sloping southwards towards the plains. They sometimes rest on Silurian rocks, and are visible in one place

FIG. 66.



FIG. 67.

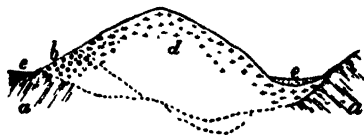
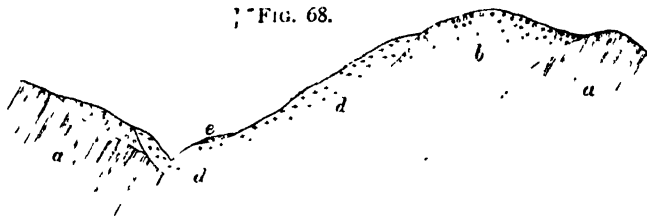


FIG. 68.



SECTIONS AT STONY CREEK.

resting on the middle Tertiary silicious rock. At the junction with the basalt, however, the surface indications so blend the characteristics of

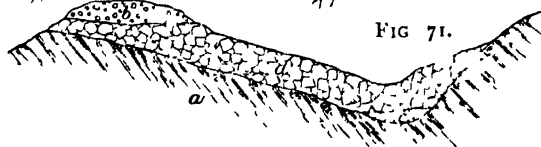
FIG. 69.



FIG. 70.



FIG. 71.



SECTIONS AT GLINMAGGIL CREEK.

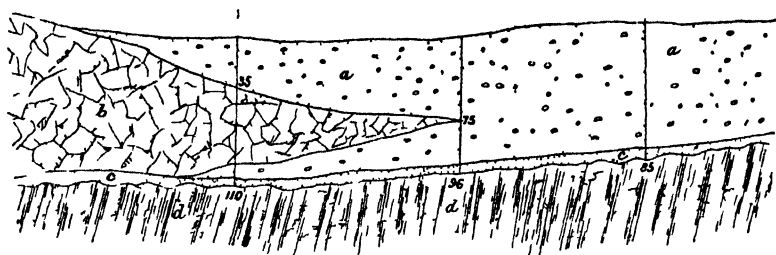
both formations that it is impossible to tell whether the gravels rest on the basalt or *vice versa*. These two possible relations are indicated in Figs. 69 and 70. The former view is adopted, because a few miles distant a clear section is found of similar gravels overlying basalt,

which is again underlaid by other gravels, as shown in Fig. 71. The reference letters are—*a*, Silurian; *b*, gravel on hills; *c*, gravel in lead; *d*, basalt.

The sedimentary deposits of the Upper Pliocene period are principally a result of the arrest, overflow, or diversion of drainage, due to the surface alteration caused by the lava-flows; and they are therefore chiefly noticed in the neighbourhood of volcanic rocks, whose limits they often hide. They form widespread layers of brown and mottled clay and sandy loam, with angular pieces of quartz and nodular masses of clay-marl, exceeding in places 70 ft. in thickness and rest indiscriminately

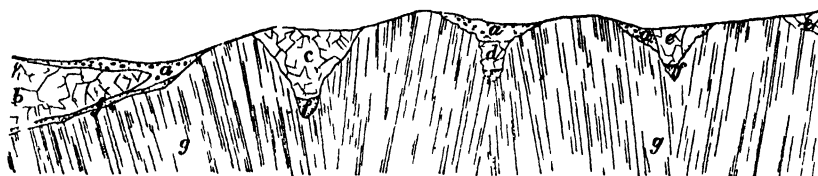
upon Palæozoic, Pliocene, and volcanic rocks. Their occurrence is well illustrated in Fig. 72:—*a*, Upper Pliocene, mottled sandy clay with

FIG. 72.



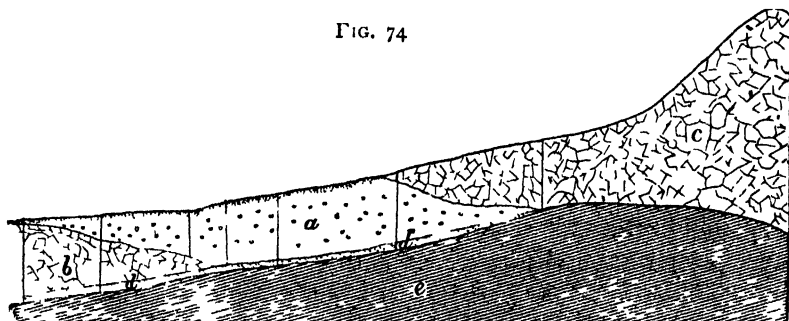
UPPER PLIOCENE LEADS

FIG. 73



SUCCESSIVE LAVA-STREAMS

FIG. 74



CLAY DRIFT BETWEEN LAVA-STREAMS.

ironstone gravel; *b*, Newer Volcanic; *c*, Middle Miocene, auriferous quartz-gravel; *d*, Palæozoic.

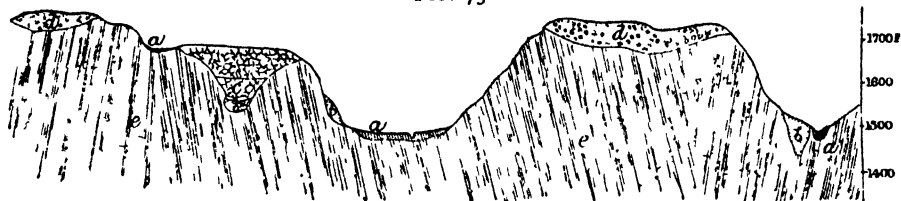
Fig. 73 is a sketch section of a number of lava streams met with in a distance of about $2\frac{1}{2}$ miles:—*a*, Upper Pliocene, clay drift; *b*, 1st lava stream; *c*, 2nd lava stream; *d*, 3rd lava stream; *e*, 4th lava stream; *f*, Middle Pliocene, gutter drift; *g*, Silurian.

Another section of about $1\frac{1}{2}$ miles of ground in the same district, is shown in Fig. 74:—*a*, Upper Pliocene, clay drift; *b*, 1st lava flow;

c, 2nd lava flow; *d*, Middle Pliocene (Lewer's lead); *e*, Silurian, along the strike of the beds.

In Fig. 75 is seen a third section, about $1\frac{3}{4}$ mile in length, showing the gold-drifts or auriferous leads of the Middle Pliocene; *a*, Post

FIG. 75



MIDDLE PLIOCENE LEADS.

Pliocene; *b*, Middle Pliocene; *c*, Newer Volcanic; *d*, Lower Pliocene; *e*, Silurian.

The lithological character of the basalt varies but little in different localities. Throughout the district there is the same granular dolerite, changing in one locality only to a glassy, more than ordinarily felspathic rock, and in another spot a quarry has exposed 18 ft. of porphyritic dolerite. Around the points of eruption prevail the usual varieties of cellular lava, scorix, volcanic ash, and breccia.

The material composing the river leads of the Middle Pliocene period, or older gold-drift, of this district, is principally a coarse, much water-worn quartz and ironstone gravel, which appears to have been derived chiefly from the denuded Older Pliocene beds. As a rule, the deepest "gutter-drift" of this gold-field is poor as compared with the auriferous "reef-wash" on the higher levels. There are evidently deposits belonging to two distinct epochs, which may be classified as "newer middle" and "older middle" Pliocene. The newer middle Pliocene occupies the channels scooped out subsequently to the deposition of the older middle drift, which rests on the flanks, or forms terraces, or has often been altogether denuded away during the newer middle epoch, as represented in Fig. 76.

A good example of the occurrence of both drifts is afforded in Carter's Company's mine, Ryrie's Creek. There two narrow gutters trend northwards towards a junction with the main Spring Hill Lead. Between these two gutters is an older channel drift, crossing the others obliquely, as seen in Fig. 77. The darker portion *b* of the plan (Fig. 77) shows the older lead, which is clayey, contains ironstone gravel, and has proved highly auriferous. The lighter portion *a* represents the deep wet gravel leads of more recent formation. They are generally poor, except where they have cut through the older leads. The older drift, south of Carter's

shaft, *c*, is probably not less than 50 ft. above the level of the gutter. The main lead and the lower part of the tributary leads are covered by basalt.

Marine deposits of the Lower Pliocene or oldest gold-drift are limited to the watershed of the lower part of Slaty Creek and the upper part of Creswick Creek, where they form coverings 4 to 60 ft. thick on many of the hills, ranging in elevation from 1700 to 1420 ft. above sea-level. The material composing these drifts consists of brown clay, with quartz gravel; coarse quartz pebbles and boulders, either loosely aggregated, or bound

FIG 76

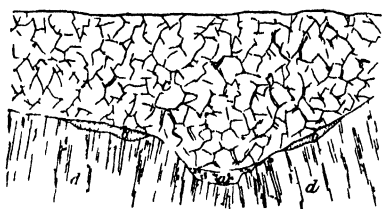
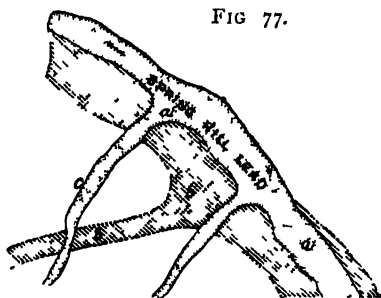


FIG 77.



RELATIONS OF NEWER AND OLDER MIDDLE PLIOCENE.

by ferruginous cement; shingle of Silurian sandstone and ironstone. No fossils have been found in them. At two hills the boulder drift has water-worn blocks of quartz over a ton in weight and upwards of 4 ft. in diameter. The gold is coarse, rounded, and generally coated with black manganese. At Hard Hill, a nugget weighing 120 oz. is said to have been found, while pieces of 5 and 10 oz. weight were numerous. The reference letters in Figs. 76 and 77 are:—*a*, Newer Middle Pliocene; *b*, Older Middle Pliocene; *c*, Newer Volcanic; *d*, Silurian.

In one instance, in New South Wales, two leads were discovered not 100 yd. apart, parallel to each other, containing exactly similar wash-dirt, but at different levels. The one at 140 to 160 ft. had no gold, whilst that at 80 to 90 ft. yielded 10 to 15 dwt. per load. From the nature of the country, it is declared that many more deep leads will be discovered when the miners acquire a more perfect knowledge of the rocks of the district, the mode of occurrence of the auriferous veins, the conditions under which they are disintegrated and decomposed, the position of the original drainage channels, how they have been silted up, and the manner in which the ancient plains and valleys have been buried beneath an alluvial deposit 50 to 150 ft. thick. The plains of the Lachlan become alluvial gold-fields wherever gold-bearing reefs or veins, however poor, are found to occupy elevated positions in their vicinity, and where on those highlands rocks of Silurian or Devonian age exhibit traces of transmutation or disturbance by eruptive trappean rocks.

In New South Wales and Victoria, the underlying geological formation is believed to control the richness of the deep leads. Selwyn says that while the leads are richly auriferous when chiefly derived from and directly resting on Silurian rocks, "they gradually cease to be so when they become underlaid by rocks newer than Silurian." This is undoubtedly true in the majority of instances, but it is not infallibly so.

In the Canadian lead, the wash-dirt often exceeds 40 ft. in depth, which appears to be due to crevasses or fissures in the limestone bed-rock. The immense deposit contains gold uniformly throughout.

At Bushman's lead, the gold from a deep lead resting on sandstone and diorite was worth 3*l.* 14*s.* 6*d.* per oz., while that obtained from intrusive dykes and small veins intersecting the rock subjacent to the drift is worth only 3*l.* 5*s.* per oz.

It is important to notice that in deep leads the deepest ground is by no means always the best. This is easily accounted for by the fact that ancient watercourses, like modern ones, may be moderately inclined at their commencement, and then become very precipitous; the gold would lodge easily in the *bed* when moderately level, but in the *banks* when precipitous. Subsequent drifts might sweep these banks, and, becoming mixed with them, scatter a portion of their gold throughout the lead.

The Nacka Nacka Creek winds in one portion of its course through narrow alluvial flats, from which rocky hills rise on either side to an elevation of 500 ft. Resting on some of the spurs, at 10 to 70 ft. above the creek, are patches of Tertiary water-worn quartz-pebble drift, the remnants of the ancient creek bed, which once descended in an unbroken course along the valley, but which has since been cut through during the erosion and deepening of the valley. Consequently, the auriferous contents of the denuded portions of the old drift have been, as it were, naturally "ground-sluced" and redistributed in the alluvial deposits along the course of the present creek. The latter should therefore be payably auriferous, seeing that the older drift has yielded up to $\frac{1}{4}$ oz. gold per load. The section shows chocolate-coloured soil, 1 ft.; sub-angular quartz-pebble drift, containing fine gold in payable quantity, 8 ft.; stiff white clay and sand, micaceous, 4 ft.; fine and coarse rounded quartz-drift, ferruginous, yielding up to $\frac{1}{2}$ oz. gold per load, 8 ft.

At Grenfell are three deep leads which were the beds of rivers that flowed in Newer Pliocene times. They are crossed by a porphyritic dyke, and are only payably auriferous for a mile below it, and not payable immediately above it, which proves the gold to be derived from the disintegration of the dyke, or veins in it.

The richness of the lead is often an indication of the proximity of reefs whence the gold was originally derived, and, conversely, the nature

of the rocks forming the hills indicates whether or not the alluvial deposits in the intervening valleys are likely to be auriferous.

In the Happy Valley lead, the bed-rock is crystalline limestone; the sinking is through clay and gravel with cement, 120 ft.; black clay, 12 ft.; clay and gravel, 10 ft.; coarse quartz-pebble wash, yielding 9 dwt. gold per load, $1\frac{1}{2}$ ft.; quartz-pebble drift and fine micaceous clay, mingled with limestone boulders, which are often coated with black oxide of manganese and brown oxide of iron, gold averaging $1\frac{1}{2}$ to 2 dwt. per load throughout the drift, 30 ft.; total $173\frac{1}{2}$ ft.

Immense deposits of richly auriferous wash-dirt have been discovered in Victoria, on the sides and ledges of deep leads: only a small proportion of the gold of the deep leads of Ballarat was taken from what is termed the lead proper—deepest channel or “gutter,”—but by far the greatest quantity of gold was subsequently found to have lodged on either side of the ancient river banks, what is familiarly termed in the deep alluvial districts “reef-wash.”

Subjoined is a sketch-section (Fig. 78) of the Wallaby diggings in Victoria. The reference letters indicate as follows:—*a*, Palæozoic;

FIG 78



SKETCH-SECTION OF WALLABY DIGGINGS.

b, Older Pliocene; *c*, Middle Pliocene; *d*, Newer Volcanic; *e*, Upper Pliocene. The gold in these leads has been derived from quartz veins traversing the ridge, and which have been exposed by the alluvial operations and since worked.

In the neighbourhood of volcanic centres, it is usual to find that the greatest extent of the latest Tertiary deposit occupies places on and around the basalt boundaries where the clay drift fills up the inequalities in the surface caused by the lava-streams. In most of the water-courses which are scooped out on, or adjacent to, the sites of Middle Pliocene leads, are found Upper Pliocene drift banks, consisting in great part of the redistributed material of the denuded older drift. Red, brown, blue, and mottled clays, with angular quartz fragments, form the matrix, with which are mixed up well-worn pebbles and boulders of quartz, and shingle of slate and sandstone. In many places, where a layer of lava

has been intersected by later erosion, as in Fig. 79, pebbles and larger masses of completely-rounded basalt form a not inconsiderable adjunct to the drift. The reference letters are :—*a*, Upper Pliocene (recent

FIG. 79

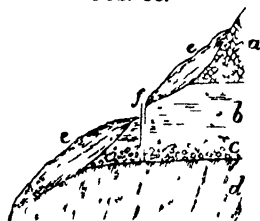


UPPER PLIOCENE DRIFT BANKS.

gold-drift); *b*, Middle Pliocene (older gold-drift); *c*, Newer Volcanic; *d*, Palæozoic.

It often happens that accumulations of *débris* make it impossible to find the exact level at which to drive into the gravel of a deep lead.

FIG. 80.

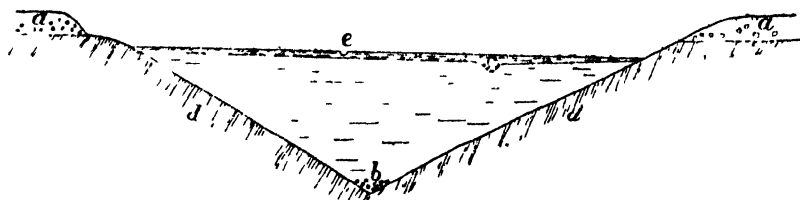


LEAD HIDDEN BY DÉBRIS.

The first object should then be to find the contact of the gravel and bed-rock *in situ* in the deepest part of the lead, and this can only be done by experimental cuttings and shafts, whose site must be selected according to the appearances of the ground, the exposures of clay and sand beds above the gravel, &c. These difficulties, and the method of dealing with them, are illustrated in the accompanying sketch-section, Fig. 80. Once the actual bed of the lead is found, subsequent operations are simple enough. The reference letters indicate :—*a*, basalt; *b*, sand, clay, &c.; *c*, gravel *in situ*; *d*, bed-rock; *e*, fallen *débris*; *f*, prospecting shaft.

Fig. 81 shows an instance of three pay-channels at different eleva-

FIG. 81.



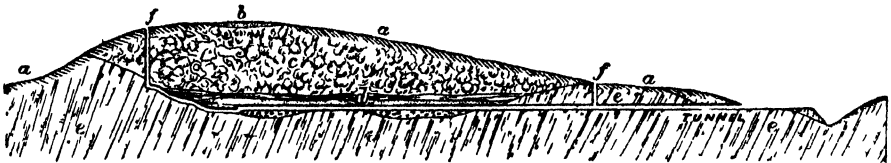
THREE PAY-CHANNELS AT DIFFERENT ELEVATIONS.

tions :—*a* is the oldest drift, worked for gold; *b*, an auriferous deep lead, 200 ft. or more in depth, resting on Silurian bed-rock, *d*; *c*, auriferous

shallow lead, about 12 ft. in depth, and resting on a false bottom; *e*, watercourse. The deep lead, in the lower part of its course, is covered by Newer Volcanic rock.

Fig. 82, on a scale of 160 ft. = 1 in., is a section showing the Tangil lead, as proved in the workings of the Tangil Gold-mining Co., about 1½ miles south of Tangil township:—*a*, surface soil; *b*, Pliocene drift above volcanic; *c*, volcanic (decomposed basalt); *d*, Miocene clays, gravels, &c., beneath volcanic; *e*, Upper Silurian; *f*, shafts.

FIG. 82.



TANGIL LEAD.

In California, the modes of occurrence of the gravel deposits are various. Sometimes they exist in well-defined ancient river-beds under a capping of basalt, which has filled the channels of the rivers of past ages; again, they appear in isolated mounds or hillocks, evidently the remains of such channels, which, being unprotected by a covering of lava, have been broken up by the action of the elements; also, in basins or flats which have received and held the wash of these disintegrating rivers; and in low, rolling hills, near the base of the Sierras, and beyond the reach of the lava-flow.

FIG. 83.

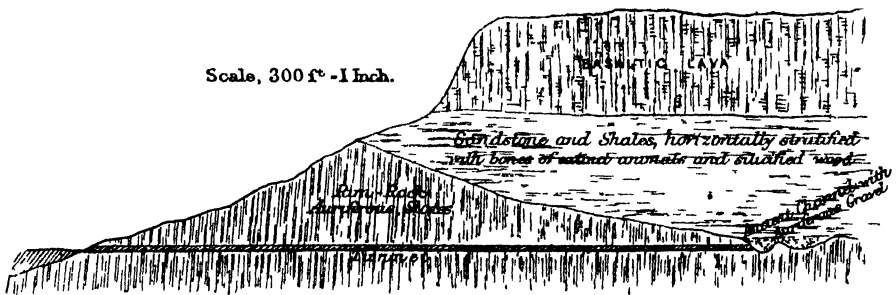


TABLE MOUNTAIN.

One of the most remarkable and important leads in America is that beneath Table Mountain, in Tuolumne county, which is represented in the accompanying section (Fig. 83).

Many auriferous gravel deposits exist in various parts of California, particularly near the limestone belt, in basins or flats. These, like the detrital matter in the foot-hills, are probably the result of secondary deposition in a recent geological epoch—the effects of the distribution by water of the ancient beds, and the denudation of the surrounding country.

The dead rivers of California, so far as known, are on the western slopes of the Sierra Nevada, and 500 to 7000 ft. above the sea-level. They are all auriferous, and have been sought for and examined. The largest and richest of them all is the "Big Blue" lead, which has been traced with certainty for a length of 65 miles, and is believed to have been followed for a distance of 110 miles. Its course is S.S.E., about 30 miles west of and parallel with the main divide of the Sierra Nevada. The elevation is about 5000 ft. above the sea at one end and 2800 at the other, giving an average fall of 33 ft. per mile. The live or modern rivers of the Sierra run at right angles to it, cutting cañons 1500 to 3000 ft. deep, and separated by ridges 3 to 6 miles apart, and as high as the cañons are deep. The Blue lead runs across these ridges at 200 to 1000 ft. below their summits, without in any way altering their appearance, or giving any clue to its whereabouts. It was discovered in detail by the same process which has revealed the existence of most deep leads—surface washing. The miners found that the modern streams were richly auriferous up to a certain point, increasing as it was approached, but ceasing when it was passed. These points were more or less in the line on the different streams, and by following up various indications, the lead was ultimately struck in several sections, and tunnelled out.

The auriferous deposit is composed of gravel, boulders, clay and sand, varying from 100 to 300 ft. in depth, and lying in strata distinguished from one another by differences in colour, in the size of the boulders and gravel, and in the number and size of the particles of gold. The predominant colour is bluish-grey, lightest at the top and deepening to a bright indigo at the bottom (hence the name). This colour is supposed by some to be derived from the bed-rock, the grinding away of which afforded the pigment to dye the quartz, &c., sulphides of iron assisting to make the colour fast. A reddish tinge in those places that have long been exposed to the air shows the presence of iron. The material composing the boulders, gravel, and sand is almost exclusively quartz. The bed may be roughly estimated at $\frac{1}{2}$ mile wide throughout, and the gravel beds some 200 ft. thick. All the pebbles, many of which weigh a ton, and some even 20 tons, are worn perfectly smooth. The gold is coarser, and contains more silver at the bottom than at the top. The finer grains are in the upper strata, and as they have a larger surface proportionately, the silver

is eaten out by the sulphurous acid which is developed in the gravel by the oxidation of pyrites. The whole deposit is formed as in existing rivers. There are banks, bars, eddies, falls, rapids, and riffles. There is much gold in the eddies, and little in the rapids. The space between the boulders is filled up with sand, clay, and gravel, which contains the gold. The bed is of slate rock, and the banks are 50 to 300 ft. high ; but there are few places where they have been examined, as the gravel has rarely or nowhere been washed away from the whole width of the bed. There is no doubt that the cause of the death of the Big Blue river was the upheaval of the Sierra Nevada range. Many of the hills pierced by the Big Blue lead are capped with basaltic lava, which covered much of the country from near the summit of the range to about 3000 ft. above the sea. It would seem that the river first silted up its bed ; that the mountain range then began to rise, and volcanoes broke out along it ; that the lava from the latter ran down and covered the land to the line of the dead river and beyond it, and finally, that the mountains rose still higher, and the streams running down their sides cut through and made deep cañons in the lava, and washed away a great portion of the ancient river deposit, scattering its gold among their own alluvia.

The upper quartz gravel, some 250 to 300 ft. thick, of which the main body of the Tancow or Butte county Table Mountain is composed, consists entirely of quartz pebbles and quartz sand, and carries very fine grains of gold. The two lower strata, however, known as the "rotten boulders" and the "blue lead" or gravel, are mixed more or less with different rocks, consisting in the former of clay-slate, and in the latter of talc-schist. Both these strata are very rich in gold, and average 5 to 25 ft. in thickness each. The "rotten boulders," so called on account of their softness, have so far been the chief resources for the miners, as they were easily reached. This stratum is said to be peculiar to the Tancow deposit.

In the preceding remarks attention has been given to leads covered with only one layer of basalt, or with no volcanic covering at all ; but in some places there are 3 and even 4 distinct layers of basalt overlying the leads. Though 3 or 4 strata of basaltic rock may be cut through, it is not certain that they always represent separate and distinct outflows. If the line of separation contains a bed of water-worn gravel, however, the disconnection of the two basaltic beds may be considered established ; but if the material intervening be simply white or red clay, with a few fragments of angular rocks, the phenomenon will more probably be due to decomposition of the basalt, for it is known that the rock decomposes *in situ* where water finds its way through it. The true bottom is never basalt, and the richest lead is almost always that lying beneath the lowest volcanic stratum.

The following tables of strata are taken from sections of formations with a plurality of basaltic beds:—

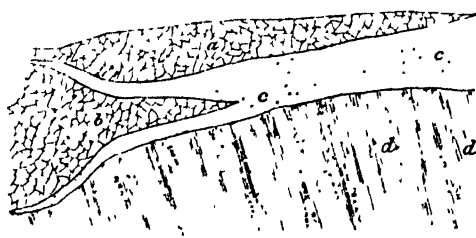
	ft.	in.
1. Three courses of basalt, separated by layers of clay ..	200	0
Black and blue clay with quartz fragments	10	0
Quartz gravel and boulders ..	8	0
Bottomed on sandstone at ..	218	0
2. Alluvial	15	0
Basalt	22	8
Clay, &c.	16	6
Basalt	72	6
Reef	0	7
	127	3
3. Alluvial	20	9
Basalt	16	0
Alluvial	17	3
Basalt	66	2
Alluvial	10	4
	130	6
4. First rock	135	0
Clay	10	0
Second rock	80	0
False bottom	15	0
Drift	10	0
Wash-dirt	6	0
	256	0

	ft.	in.
5. Clay	11	0
Basalt	28	0
Red clay	8	0
Black clay (extremely light) ..	15	0
Basalt	92	0
Black clay	3	0
Green stuff, like reef	9	0
Sandy fine drift	4	0
Basalt	12	0
	182	0
6. Surface soil	9	0
Rotten basalt	20	0
Massive blocks of basalt	14	0
Red sandy clay	8	0
Basalt	51	0
Gravel and clay	13	0
	115	0
7. Basalt	36	0
Red clay	6	0
Basalt	59	0
Clay	4	0
	105	0

8. Surface earth, 7 ft. ; basalt, 130 ft. ; stiff clay, 18 ft. ; basalt, 29 ft. ; mixed clay, gravel and drift, 83 ft. ; wash-dirt, 5 ft. ; total, 272 ft. Width of gutter, 100 to 250 ft. ; depth of wash-dirt, 4 to 7 ft. The wash-dirt was a dark-coloured coarse gravel, mixed with clay, containing large quartz boulders, charcoal, and decayed wood.

9. Basalt, 160 ft. ; very hard sandstone conglomerate, 20 ft. ; second layer of basalt, containing a large quantity of water ; layer of grey clay ; and a layer of black clay, so largely composed of vegetable matter as to resemble coal in appearance and structure. On the Ballarat gold-field, this black clay nearly always overlies the "gutter," and its presence used to be regarded by the miners as a certain indication that the shaft would bottom in or near the gutter.

FIG. 84.



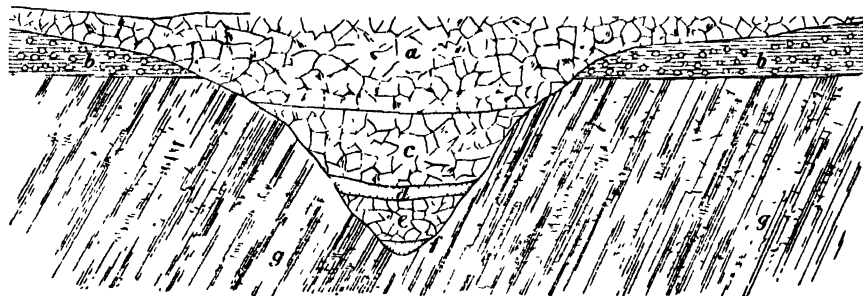
LAVA-STREAMS AT HAWKINS' LEAD.

Fig. 84 is a sketch-section showing the occurrence of lava-streams at the Hawkins' lead:—

a, 1st volcanic rock ; *b*, 2nd rock ; *c*, clay ; *d*, schist. Fig. 85 is an ideal section showing the position of the Durham lead under the Mount Mercer

plains:—*a*, Mount Mercer lava-flow; *b*, Older Pliocene; *c*, Durham lead (2nd rock); *d*, middle drift; *e*, Durham lead (3rd rock); *f*, lower drift; *g*, Silurian bed-rock.

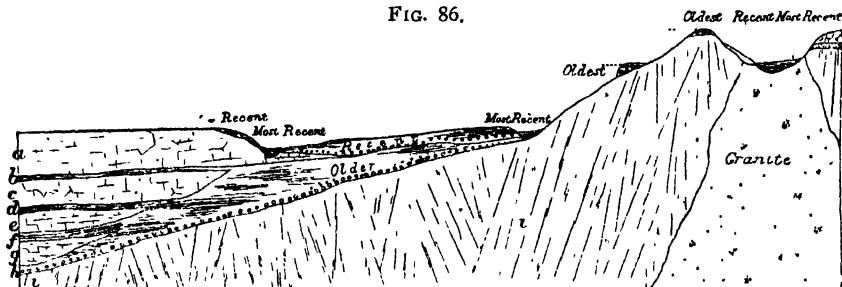
FIG. 85.



DURHAM LEAD UNDER MOUNT MERCER PLAINS.

Fig. 86 is a sketch-section showing the occurrence of the strata in the celebrated Ballarat district, Victoria:—*a*, 1st rock or lava; *b*, clay; *c*, 2nd rock; *d*, clay; *e*, 3rd rock; *f*, clay, sand, &c.; *g*, 4th rock; *h*, clay, sand, and gravel (gutter drift); *i*, Lower Silurian bed-rock.

FIG. 86.



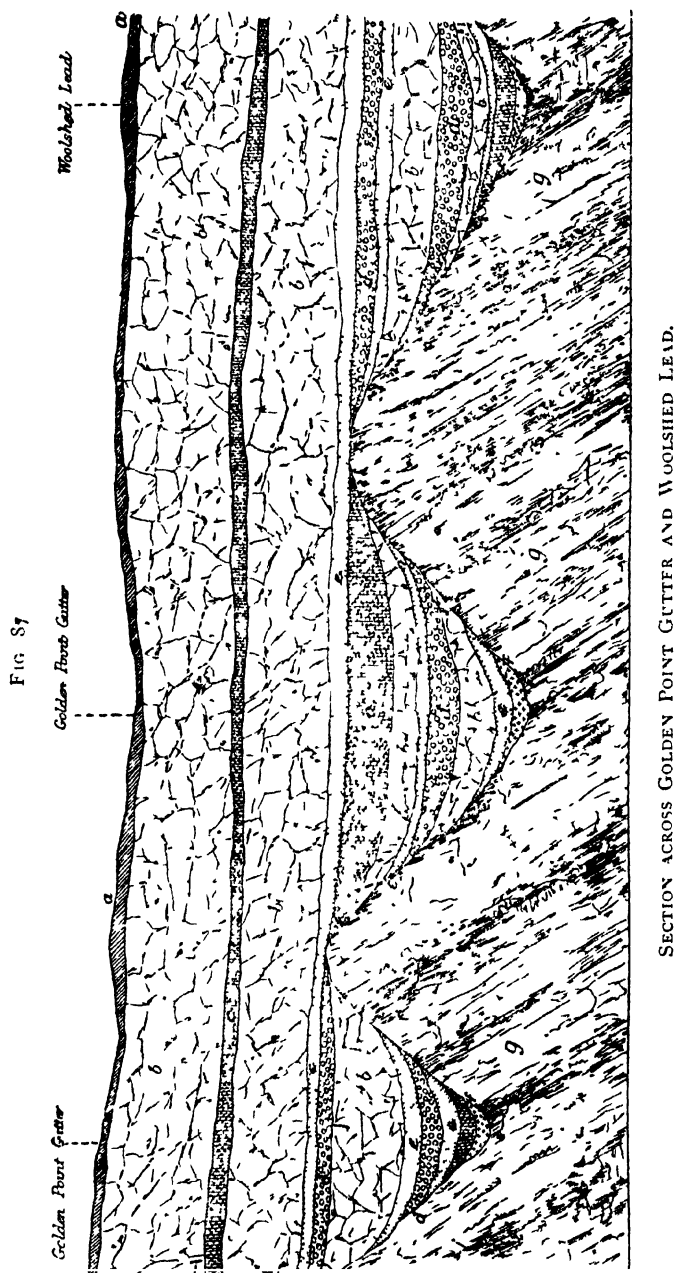
SKETCH-SECTION OF STRATA AT BALLARAT.

Fig. 87 is a section across the two arms of the lead known as the Golden Point Gutter, and the Woolshed lead, on the Ballarat gold-field:—*a*, surface soil; *b*, basalt; *c*, black or red clays; *d*, sandy drift; *e*, light-coloured clays; *f*, auriferous drift; *g*, Palæozoic bed-rock.

Special features or peculiarities are constantly being noted, and should receive the most careful attention. Some will be found scattered already throughout this chapter, but others remain to be mentioned.

At Ballarat, the reef was generally pipe-clay; but in some places sandstone bottom was found: in the latter case, the lead was much poorer than in the former, that is to say, it had not offered such great obstruction to the passage of the gold down stream.

It is commonly believed that where dead-river beds meet, the wash-dirt must necessarily be rich and abundant, and it is usual for miners to



rush to secure ground where leads are supposed to unite. Where all the leads tending to the junction are known to be rich, these expectations

are not likely to be disappointed ; but, in most instances, the miner should carefully ascertain the character of the main leads and their tributaries. A very rich lead is often rendered worthless by the influx from a tributary, of great quantities of water charged with *non*-auriferous sands and gravels, derived from a large drainage area barren of gold-bearing rocks. The greater force of united currents, too, will scatter over a wide surface and through a great mass that which was before confined within smaller limits.

Where the Palæozoic rocks are exposed by the denudations which have swept away the basaltic lavas, it will be possible to reach the gutters of the deep leads with exceptional ease.

An instance has been known of a deep lead trending in the opposite direction to the flow of the modern stream above it, that is to say, that the dead river unmistakably ran northwards, while the living one flows southwards. This is, of course, due to alterations in the configuration of the country.

In trying to sink through a false bottom on one claim, there was discovered, at a depth of 90 ft., a vein of wash-dirt almost perpendicular, or dipping slightly, like a quartz reef.

There is a peculiarity noticeable about the wash-dirt in a lead in Montana, that, wherever it is composed of quartzite and quartzite slate it pays well, but where it is largely made up of granitic rocks, little or no gold is found in it.

Where the lead becomes very narrow, dips fast, and is enclosed between steep walls, the gold will be very sparingly distributed in holes and behind ridges, and will be coarse in size.

On the Creswick gold-field, in Victoria, it is observed that, as a rule the deepest gutter drifts (Newer Middle Pliocene) are poor as compared with the reef-washes (Older Middle Pliocene) of the higher levels. This seems to show that the older drainage courses have been the receptacles of gold derived from a much greater denudation of the surrounding country than those of the more recent period which now occupy the lower levels.

The occurrence of auriferous drifts between layers of volcanic rock has previously been noted in a few instances ; but it is rare that the gold exists there in very payable quantity. Krausé, however, draws attention to a specific place where such a deposit is yielding $\frac{1}{2}$ dwt. of gold to every machine-ful of wash-dirt.

Modes of working.—It is deemed unnecessary to occupy space here with a discussion of boring, shafting, tunnelling, and other matters incidental to mining in general, as such subjects have been already sufficiently well treated in other works, notably André's treatises on Mining Machinery, and on Coal Mining. Special attention may be directed to

his description of Chaudron's shaft-sinking machine, a Belgian invention of great value. There remain for consideration a number of points, more or less important, having a special bearing upon the extraction of gold from deep leads.

Drifting.—The manner in which the pay-dirt is worked when it has been reached by a shaft varies according to circumstances. Perhaps the most general plan is that known as "drifting." This consists in working out the auriferous gravel by a system of tunnels and galleries, and raising it to the surface so as to run it through sluices or cradles for extracting the gold. In this case, nothing but the gold-bearing earth or gravel is removed. The main tunnel is carried on upstream and the galleries are run to either bank (or rim). The gravel is worked out, and the galleries are generally allowed to fall in as the work advances, the main tunnel being secured by timbering. Raymond says that the depth of gravel which it is found will pay to work by this system is 2 to 5 ft., but it is obviously impossible to lay down any hard-and-fast rule.

In places where the beds of the modern streams are lower than the level of the wash-dirt, horizontal tunnels will generally, not always, be found most economical. These must be driven with such a slope upwards from the mouth that the water accumulating in the tunnel will naturally *find an outlet by running along the bottom of the tunnel*. In most cases, *the tunnel requires some sort of timbering to support the sides and roof*. It is quite impossible to give the slightest clue to the cost of tunnelling, as it depends upon so many conditions, which vary in every instance. The wetness or dryness of the ground; the degree of hardness of the rock; the facilities for, or difficulties in the way of, procuring suitable timber for props, cap-pieces, and slabs, where necessary: all tend to increase or decrease the cost, just as the circumstances are favourable or otherwise.

"When the auriferous gravels contained in the ancient river-channels are covered over to any great depth by the volcanic eruptions of past geological periods; or where the pay-gravel is concentrated in a layer, from 2 to 8 ft. thick, directly on the bed-rock under a very thick deposit of either barren or very poor gravel which could not be profitably mined by the hydraulic process; or where, from scarcity of water, or difficulty of getting dump room, owing to lack of fall of the ground below the mine, it would be impracticable to move the whole of a deep placer deposit; the gravel is mined, and the method of working adopted in California, is called 'drifting,' and the claims are called 'drift-claims.' The volcanic rock or gravel above the gold deposit cannot be moved, and the ancient channels, if reached at all, must be reached by tunnels run through the side of the rim-rock at sufficient depth to reach the deposit and make it possible to take out the pay in the ancient channel. Drifting is never done where hydraulicing is possible." (Egleston).

The subjoined remarks are condensed from a valuable paper on "Drift Mining," by Prof. T. Eggleston. The industry is followed on a grand scale in California, and no authority is better qualified to give an account of it. The greatest judgment of the engineer is required to settle what is advisable in order to successfully open a given piece of ground. The probable width of the channel can only be guessed at. Its depth can be ascertained by prospecting. As a general rule, it may be said that an acquaintance with the known channels which have been worked out in the locality is the best guide to work on. Long experience has shown that the highest slope of a channel will not usually exceed 30° . Channels vary from 40 or 50 ft. upwards in width, and will average about 250 ft., but as a general thing will not exceed 600 to 700 ft.

A rough kind of drifting was formerly employed to a small extent in working bars on rivers, which were loose and easily mined, but afforded little material to work upon. The plan was to follow the pay leads under the banks, supporting the excavations by boulders, and working for short distances only, leaving holes rather than drifts, which soon caved. This method, which could not be worked on any system, was called "Coyoting."

The amount of auriferous gravel existing in drift-mining claims, is not less than on those worked by hydraulic; but the poorer upper gravel cannot be profitably extracted in drifting on account of its position, and must, therefore, be left, only the rich bed-rock section being worked. The ancient rivers differed from the modern ones only in being on a very much larger scale. There were in these rivers, as in those of to-day, long barren stretches, as well as pockets or crevices of great richness. In modern streams, the barren places are found from external indications without much difficulty, while in the ancient streams, covered with lavas, they must be sought for almost at haphazard. It often happened in these streams, as in modern ones, that there was more than one channel; they are not of necessity equally wide, but they must not be passed over if they are there, as the expense of working is nearly the same whether there are one, two or three. The problem of drift-mining is to find the "channels," to open them up with the least possible expenditure, and then to work them with the least possible cost. It is not always possible to find any surface indications of an old "channel." Some of the best mines worked in California at the present time, and nearly all those worked out and abandoned, were found by following the surface placers up to the line of the disappearance of the gravel and gold. At about this point there is generally a "channel" opening, and here prospecting must be done. Sometimes a depression in the line or under edge of the country rock indicates an ancient "channel" and the probable presence

of gold, but this cannot always be relied upon. Gravels, however, are not necessarily confined to a definite "channel," as in many cases the ancient streams, like the modern ones, overflowed their beds, carrying gravels and sands with them wherever they did overflow. Nearly all these sands contain some gold.

Auriferous gravels sometimes cover the whole surface of the country without any apparent defined channel. There is also the greatest variation in the consistency of the volcanic rocks which cover them. Some of them are extremely hard, others thoroughly decomposed and easily separated. The channels are subject to a certain number of irregularities and disturbances, and are sometimes temporarily cut off entirely. These breaks are often called by the miners "lava flows," but as a general thing they are owing to the displacements of the bed-rock by the weight of the gravel above, or else they are disturbances caused by a movement of the pipe-clay. It is not an uncommon thing for the bed-rock also to swell, and in some instances it is extremely difficult to keep the tunnels open on account of this. Occasionally the bed-rock appears to rise and then to descend, and in many cases this has undoubtedly been induced by the additional pressure caused by the erosion of large bodies of ground, leaving great spaces between the ends of the "lava flows," thus causing the bed-rock to bulge. When the bulge occurs in two places at some distance apart, it gives the appearance of a basin; and when only two sides of it are found, this may lead the inexperienced miner to suppose that his claim is worked out in that direction.

The gold in these deposits varies greatly in size. It occurs from fine float dust up to nuggets from 40 to 60 oz. in weight. Most of it, however, lies in a few feet of the gravel on the bed-rock, while the whole of the upper gravel is too poor to work. Some few of the present hydraulic mines were originally commenced as drift claims, when they sometimes yielded a larger profit than they now do as hydraulic claims.

For prospecting purposes, tunnelling is generally too costly. When the capping is not expected to much exceed 150 ft., a vertical prospecting shaft is commonly adopted. This shaft is made circular, and only affords room for one workman at a time. It requires little timbering and no hoisting machinery beyond a windlass. Its cost may vary from 10s. to 50s. a foot, according to the hardness of the gravel. The shaft may with advantage be preceded by test borings. Drifts can be run from the bottom of the shaft in all directions on the bed-rock. Sometimes shaft prospecting may be seriously interfered with by flooding.

Exploration having determined the probable location of the channel and proved that the claim contains pay gravel, a point is selected where

a shaft is sunk to the bed-rock, and drifts are run in various directions from the bottom. The position of the channel having been determined in this way, a tunnel is run through the rim-rock so as to intersect the bottom of the channel in the shortest possible distance, in order to avoid a great length of tunnel; and when this has been done the mine is laid out with reference to its extent.

If what is supposed to be the entrance of a channel is found, the tunnel is commenced some distance below the seam of "rim-gravel," as it is called, and run on light grade until it comes into the gravel or is supposed to be beyond the rim-rock, at which point a winze is sunk to the surface of the bed-rock in order to reach the gravel. This winze is usually at right angles to the direction of the channel and is inclined considerably. The course, width, and grade of the channel are determined as near as can be with the least amount of work.

To determine the centre of the channel, the surface of the bed-rock is followed until it is found to rise and fall upon two opposite sides. But it does not always follow that because two inclines in opposite directions have been formed that the end of the pay dirt has been reached, for it sometimes happens that the channel is divided, or that there are elevations and depressions in it. An examination must be made to ascertain whether it is really the rim, or only an accidental elevation or depression towards the sides of the channel. The main tunnel is carried up the grade of the ancient stream, and gangways are run towards each bank. The gravel is taken out in cars and washed in sluices. The main tunnel is heavily timbered, so as to be always protected; but the side drifts are temporary, and, when the "pay" has been removed, are allowed to cave in.

The depth of gravel that has been found to pay for treatment by this system is from 3 to 5 ft., and averaging not much over 3 ft. This includes 3 to 6 in. of the bed-rock, which is always picked down and washed with the gravel, as the lower part of the deposits, including the bed-rock, is always the richest. Occasionally a "false bed-rock" may occur, caused by volcanic matter filling up an ancient stream, and being followed subsequently by the waters flowing very nearly on the course of the former bed.

An incline in the cement in one claim led to the discovery of another channel running north-east and south-west 80 ft. lower than the original channel, and containing an entirely different kind of gravel, consisting of granite pebbles and boulders instead of quartz. The character of the gold is also different, being flakey, nearly uniform, and of slightly finer quality. Generally, if there are several gutters, the inclined rock supposed to be the "rim" will turn down, thus suggesting further exploration.

Wherever practicable, a drift mine is always worked through a tunnel run into it from the hill side or from a ravine, so that its bottom will be either on or little below the bed-rock, with only just enough inclination to ensure drainage. On the location and construction of this tunnel the success of the mine depends. When it is impossible to reach the gravel by a tunnel, or where the latter would be very long or expensive, recourse is had to vertical or inclined shafts. The first cost of reaching the gravel by a shaft is less than by a tunnel; but for taking out the gravel the tunnel is desirable, for in working by shaft all the gravel and mine water must be raised to the surface, entailing expensive hoisting and pumping machinery. Practically it costs about as much to bring the gravel from the point where it is mined, to the foot of the shaft, and load it into the cage or bucket, as to take it out in cars through a tunnel 1500 to 3000 ft. in length. Therefore a claim which may pay well for tunnel working may cause a loss in shaft working.

The tunnel must be securely and heavily timbered. After reaching the channel, it is continued up stream, as nearly as possible in the centre of the channel. The gravel overhead is supported by timbering until the pay gravel for a considerable area is removed. In order to prevent its crushing in the main tunnel, the timbering is made very strong, and pillars are left in the ground 30 ft. or more on either side of the main tunnel. When the limit of the claim has been reached, these pillars are robbed. It will generally be necessary, in order to obtain sufficient dumping ground, to run an incline on the surface, from the mouth of the tunnel to the place where the dirt is treated. The inclination of this slope is usually 8 ft. in 100. If the material is to be stamped, it must be delivered into bins. If the dirt is soft, the machine ends in a shute, which allows the dirt to slide into the space where it is to be washed.

Sometimes the mistake is made that the tunnel is not run low enough, and, instead of remaining in the bed-rock comes out into the gravel. Should the bed-rock pitch off in the direction in which the work is being prosecuted, it may be necessary to run a new tunnel lower down. This will depend principally on the amount of water in the mine. If the tunnel has been run too low, the gravel from the drifts will have to be dumped into a shute to be loaded into the cars below. If too high, an incline will have to be run to the bed rock and the gravel hoisted. When the expense of a regular tramway is not justified, the ore is brought in large buckets, which are turned on their side and hoisted on two pieces of timber laid far enough apart to catch the sides of the bucket.

Occasionally the conditions are such as will admit of the tunnel being run entirely in the gravel, but these are very exceptional. From the centre of the channel the tunnel is run in the bed-rock on the line of the

channel as the developments in the mine show what the course is to be. To keep the tunnel in the bed-rock, its entrance must be much lower than the channel bed. What this depth must be depends upon the grade of the channel, its flow, the grade of the tunnel, and the length of the ground it is intended to work. All these, excepting the grade of the tunnel, can at the commencement of the work be determined approximately only. It is better to have a tunnel much too low than a little too high, so a very liberal allowance is generally made to cover possible errors of calculation from uncertain information. The dimensions of main tunnel formerly adopted were : height, $6\frac{1}{2}$ ft. ; width, when not timbered, $5\frac{1}{2}$ ft. at the bottom, narrowing towards the top. The posts are 10 to 20 in. in diameter, and for special purposes a diameter of 2 ft. is not uncommon. Caps are generally 12 in. and upwards. The original method of making a nearly square tunnel has been abandoned in most of the large mines, and the system is now to make the main tunnel, for a single track, 7 ft. wide on the bottom, $6\frac{1}{2}$ to 7 ft. high, and $3\frac{1}{2}$ ft. wide at top, these measurements being in the clear. When double tracks are used, the width is 12 ft. on the bottom. The object of placing the posts slanting is to prevent rapid crowding at the bottom in soft ground, which, with the posts set perpendicular, would cause very great delay and expense for repairs.

The tunnel is run in on a slight incline, so as to secure drainage and make the transportation of the gravel easier. Grades varying from $\frac{1}{4}$ to $\frac{1}{2}$ in. in the rod are the lowest, often reaching to 3 or 4 in. In general, the lighter the grade consistent with proper drainage, the better. As the bottom of the channel, when reached, is never regular, and the grade of the tunnel must be, it will sometimes cut into the bed-rock, and sometimes be in the gravel, but these irregularities do not affect the general working. When the rock in the tunnel has a tendency to cave in, to slide or to swell, it must be heavily timbered. If necessary, lagging is put on the top and on the sides. Timber which has been roughly squared with an axe is generally used. All that is to be used in the mine is generally cut, dressed, and delivered by contract at 8 to 12 cents. (4d. to 6d.) for the single stick. Lagging costs usually from \$20 to \$40 (£4 to £8) per thousand. The wood used is generally the pine, spruce, fir, or cedar, which covers the ground of the claim, or grows near by. Usually, enough can be obtained from the surface of the claim to supply all the wants of the mine. Men are employed by contract during the summer cutting timber and lagging.

Obviously the cost of running the main tunnel and sinking the shafts in the bed-rock must largely depend on the character and hardness of the rock. In slate or other soft rock it may cost as low as 25s. per foot, including the timbering and lagging. Harder rocks may cost up to 4l.

In blasting the harder varieties of rock, giant powder is used. Black powder is found most efficient in serpentines and similar rocks. The most economical way to run a tunnel is by contract.

Below is a statement of the cost of tunnelling done at Red Point, Placer County, California, by the Golden River Mining Co., up to Feb. 1st, 1887. It is the intention of the company to tap and work an ancient gravel channel which is known to exist under the ridge forming the upper portion of the Forest Hill divide. For this purpose a tunnel was located in Red Point Cañon, about $1\frac{1}{4}$ miles above the town of Damascus. An air-compressor is located 200 ft. vertical above and about 300 ft. distant from the mouth of the tunnel, the air being conveyed through a 3-in. pipe, which in the tunnel has valves and blow-outs every 500 ft. for the purpose of ventilation. There are no other pipes in the tunnel, except a 1-in. water-pipe for drilling use.

Work on the tunnel proper commenced on the 2nd of July, 1886, and 108 ft. were run by hand with an average force of 6 men per day. The compressor was started on the 5th of August, and on the 31st of January, 1887, the tunnel had reached a length of 1552 ft. The last 1444 ft. were run with an average force of $20\frac{1}{2}$ men per day.

The two largest runs made for two consecutive weeks were 71 ft. for the week ending August 28th (6 days' work) and 84 ft. for the week ending September 4th, or respectively 11.82 and 12 ft. per day.

The tunnel runs diagonally across the strike of the rocks, which consist of alternate strata of slate, impregnated with iron pyrites, diorite, and masses of white barren quartz.

The regular force of men employed (when full) consists of

15 miners..	..	..	working	8 hours per day.
2 engineers ..	..	..	12	" "
2 drivers..	..	..	12	" "
2 blacksmiths..	..	..	10	" "
1 timberman ..	..	..	10	" "

divided into 3 shifts and working 2 drills on columns. The blacksmiths and timberman only worked in daytime, unless on account of hard rock it was necessary for the blacksmiths to work extra time.

COST OF SURFACE IMPROVEMENTS, PLANT AND TUNNEL.

	Total Cost.	
	\$	£
Road, 6500 ft. long; average grade 1 ft. in 10. Commenced May 26, 1886; completed July 1, 1886; average force of men per day, 9.2.		
Including surveys and powder	963.00	192 12
Yards, dump and trails. Commenced June 21; completed July 31; average force of men, 5.6. Total cost, including crib-work, timber and powder	508.05	101 11
Boarding-house, office, blacksmith shop, stable, powder-house, woodshed, framing sheds, snow-sheds, &c.	2,310.10	462 0
Carried forward	\$3,781.15	£756 3

	Total Cost.	
	\$	£ s.
Brought forward	3,781·15	756 3
Water-works. Long dam, 2-in. pipe line 2800 ft. long, in ditch and covered. Commenced May 30, and completed July 30; average force of men per day, 2·74. Total cost, including surveys, powder and pipe	604·94	120 16
Air-compressor erection. Commenced July 1; completed with pipe line to tunnel July 30; average force of men per day, 5. Total cost of labour	970·00	194 0
One straight-line compressor, 16 × 16 × 24; one 34 × 16 steel boiler, complete; air-tank; pump, 3 × 3½; eclipse drills and columns; pipe connections, extras, freight and building, 30 × 40 ..	7,819·86	1564 5
8 iron cars	1,200·00	240 0
2 tunnel horses, 2 team horses, and buckboard	705·00	141 0
Total cost of plant, &c., &c.	\$15,080·95	£3016 4

ACTUAL COST OF 1552 FT. OF TUNNEL 7 × 8, EXCLUSIVE OF MANAGEMENT, UP TO
FEBRUARY 1, 1887.

	Total.		Cost per running foot.		
	\$	£ s.	\$	£	s. d.
Total labour (pay-roll), including 100 ft. of timbering	11,418·87	2283 15	7·36	1 10	9½
Powder. 10,567 lb. Giant No. 2, and 325 lb. No. 1, at 26½ c., 41½ c., 10 per cent. off	2,641·64	528 5	1·70	0 7	1
Fuse. 39,650 ft. at \$5·50 per mille, and caps \$45, delivered	263·07	52 16	0·17	0 0	8½
Wood. 402 cords, at \$2·75	1,105·50	221 0	0·71	0 2	11½
Charcoal. 1604 bushels, at 20 c.	320·80	64 0	0·21	0 0	10½
Candles. 1760 lb. at 16½ c.	290·40	58 0	0·19	0 0	9½
Foot planks and ties. 7355 ft. of lumber, at 20 per mille	147·10	29 10	0·09	0 0	4½
Timbers. 43 sets, at 60 c. per running foot	46·76	9 6	0·03	0 0	1½
Steel rails. 16,640 lb. at \$60 per ton	510·00	102 0	0·33	0 1	4½
Air and water pipes, 3 and 1 in., at 18 c. and 5½ c. per ft.	521·86	104 4	0·35	0 1	5½
Horse feed	281·25	56 0	0·18	0 0	9
Materials, steel, oil, tools, &c.	693·00	138 0	0·45	0 2	0½
Freights, at 1·25 per 100 lb.	1,000·00	201 0	0·64	0 1	4
Totals	\$19,239·85	£3847 16	\$12·40	£2 11	8

When the main tunnel is run underneath the centre of the channel, a winze or an incline, the winze being preferable, is made through the bed-rock to the gravel. If it is not intended to continue the work any farther, an incline is sometimes preferred, especially if the timbers and other materials used in the mine are to be taken through it. They are always constructed in two compartments: one for the gravel; the other, containing the pipe, for drainage and the ladder for the men.

Drainage must be done by natural means if possible. Where there is a bed-rock tunnel below the workings, it is done through the lower tunnel. Usually there is much water in these channels. If the quantity to be handled is considerable, the water which collects in the mine-workings is

conducted to a main sluice (when the tunnel is in the gravel), or, when the tunnel is in the bed-rock, to the several shafts, and, passing through them, runs either by a trough made under the tramway, or by a ditch at one side, to a large reservoir tank at the entrance of the tunnel, in which it is stored to be used for washing the gravel. A pump of considerable capacity is almost invariably required in such mines when they are worked with vertical or inclined shafts. The flow of water usually increases as the mine is more completely opened, and is generally rather beyond, than within, the control of such pumping machinery as can usually be had. It sometimes happens that the tunnel, running too high, will not permit of natural drainage, being perhaps 20 ft. higher than the bed-rock of the channel. In this case, the running of a lower tunnel can be obviated by collecting the water in a sump close to the tunnel, and emptying it by means of a siphon. In some claims "hurdy-gurdy" wheels are used. The power developed usually ranges from 5 to 20 H.P. for each wheel. Sometimes small overshot wheels are used for pumping, for driving air-compressors, and for motive power in the cement mills; but the wheels are usually of the hurdy-gurdy type. The same power is used for hoisting.

At the Mountain Gate mine, instead of allowing the waters to run into the lower tunnel, they are collected in a reservoir cut out of the bed-rock at the inner end of the old workings, 6500 ft. from the outlet of the tunnel. From this tank there is a fall of 40 ft. to the lower tunnel. An overshot wheel 30 ft. in diameter is run from this tank, and furnishes power sufficient to pump all the water of the lower tunnel and raise the cars of gravel.

The method of extraction is not at all unsimilar to that which is employed in ordinary coal mining. When the pay gravel is reached, it is cross cut from rim to rim, gangways are run at right angles, and the ground is stoped out in breasts. The main tunnel is always run ahead of the gangways. Pillars 20 to 30 ft. wide are left on each side to support it. It should follow as nearly as practicable the lowest point or gutter of the channel. The breasts are supported by temporary posts, or by piling up the large boulders and rejected material which have been found in the course of mining. As each breast is worked out, the ground is abandoned and left to cave. Occasionally the swelling of the bed-rock below hastens the closing of the worked out chambers.

The main gallery is generally 10 or 12 ft. wide on the bottom, 8 or 9 ft. high, and 3 or 4 ft. wide on the top. The breasts are usually 3 to 5 ft. in height. When the gravel must be carried to the surface by vertical or inclined shafts, the kind of motive power to be used depends on circumstances. If there is but little gravel, it may be raised by hand; if a large quantity, mule, horse, or steam-power is used, or water-power, in the

shape of a hurdy-gurdy wheel, when water can be had. When there are no shafts or inclines, but little power is required, as the grade of the tunnel is usually sufficient to run out a train of two or three cars with one man at the brakes. When the quantity of gravel is very large, mine locomotives must be used. The cars with the timber and tools in them are returned by mules except when mine locomotives are employed.

The ground is broken up, working forward from gangway to gangway, taking out all that will pay, which sometimes is as much as 25 to 30 ft. high. Sometimes the largest part of the gold is found within 1 ft. of the bed-rock. In such a case, only just enough gravel is removed to enable the miners to work. This will make the workings 3 to $3\frac{1}{2}$ ft. high. Generally 4 to 6 ft. is rich enough to pay. The cases where it will pay to take more than this are exceptional. Sometimes at a comparatively small distance above (20-25 ft.) there may be another stratum of gravel that will pay to drift. Three such pay streaks have been discovered and worked in some claims, necessitating a separate tunnel. The roof is supported with ordinary posts and caps until the breast is worked out, when they are removed and the ground is allowed to cave. The gravel is thrown by the drifters as near to the gangway as possible, where it is loaded into buckets or cars by the carmen, and pushed to the foot of the incline to be hoisted. The men also run in the empty cars on their return. When the distance increases, this work should be done by mules or by a locomotive. When the mine is in regular work, after opening, transportation by men is too expensive. It was found at the Bald Mountain mine that tramping by men cost 21 cents ($10\frac{1}{2}d.$) the load, by mules 9 cents ($4\frac{1}{2}d.$), and by locomotive $4\frac{3}{4}$ cents ($2\frac{3}{4}d.$) per load. The rails used weigh 16 lb. to the yard, and are ordinary T rails. The gauge of the track is 18 in. The cars are the ordinary box car, which holds $\frac{3}{4}$ ton of gravel, or platform cars made as low as possible, so that the men can easily roll large rocks on to them. Whatever rocks are taken from the mine are, as a rule, taken out entire. Occasionally one is found to be too large. It is then blasted by laying a piece of giant powder on top of it, covering it with sand, and then exploding it. This is, however, only done in the opening of the mine. As soon as the breasts are well opened, and the bed rock is exposed and well cleaned, the rocks taken from the face are piled into the empty spaces and left there.

The gravel is sometimes quite tight and requires blasting. Holes are made 20 in. deep, loaded according to the toughness of the gravel with from $\frac{1}{2}$ to 2 sticks of giant powder. When it is necessary to enter the bed-rock, rock drills are used. The main tunnel is always run by gangs of picked men, who work the tunnel in 8-hour shifts. It is kept as far ahead as it can be, and as near the centre of the channel as possible. The gangways running off to the rims show the deviation from the centre

of the channel where there is any. About four car loads of gravel to a man is broken. This includes all the men at work in the mine. It varies somewhat with the amount of gangway work, the number of large rocks found in the gravel, and the amount of bed-rock to be cut up.

The main and cross drifts are run in the gravel on the surface of the bed-rock if it is hard and contains no gold. If the bed-rock is soft, and does contain gold, 8 or 10 in., sometimes 2 ft. of it, are taken up. The gravel from different drifts is washed separately, in order to determine the probable value of the adjacent ground, which is divided into squares for the purpose of prospecting. Where there is hardly any uniformity in the distribution of the gold in the gravel this is not done to any extent. In estimating whether a certain patch of ground will pay, the gross yield, the amount of timber or lagging, and the hardness of the gravel must all be taken into consideration. No general rule can be given as to what gravel will pay and what will not. The degree of richness which will make mining profitable must be determined in each special case. In working the gravel, only the finer portions are taken out to be washed. As soon as a gangway has been worked in 7 or 8 ft., the track is advanced to the heading, so as to bring the cars close up to the face. When this is again worked off 7 or 8 ft. in depth, the track is again moved, and the cars are brought up again. When 10 ft. or more of gravel is mined, the breasting is done in two benches. Very long timbers are used; and if the gravel is soft and liable to cave, great care must be taken to place the timbers and lagging properly. Except where the gravel is very hard and cemented, large amounts of timber are required. In many mines, in addition to timbers and caps placed closely together, it is necessary to use top lagging, driving the latter ahead of the timbers, and close up to the face as the gravel is taken away. Blasting is only required in extremely hard and cemented gravels. In most mines, the ground is so soft that it can be easily worked with a pick. As far as practicable, the gravel is broken with a pick, as it is difficult to use powder to advantage. Where the gravel is quite soft, it may be washed by projecting a stream of water under pressure upon the loosened dirt.

The cars and track used in gravel mines when they discharge through shafts to a lower rock tunnel are somewhat different in construction from those used for carrying the gravel to the dump. The track in the workings has to be moved frequently, and hence is often made in sections. Sometimes wooden rails are used, faced with strap-iron 2 in. wide by $\frac{1}{4}$, $\frac{3}{8}$, or $\frac{1}{2}$ in. in thickness. 1 in. thick is found to be best. Wooden rails, whether strapped or not, are being rapidly replaced by iron or steel T rails, weighing 16 to 42 lb. to the yard. Iron or steel rails have a great advantage over wooden, because, when the ground heaves, as it very often does, the rails rise in long swells, instead of making sharp bends as with wood. In the

main tunnel, and in all the drifts which are permanently used, the tracks should be laid with great care. The wear and tear of the rails is severe, and frequent repairs are necessary. The gauge of the track varies from 16 to 22 in. Mine cars are generally $4\frac{1}{2}$ to 5 ft. long, 24 in. wide and 20 to 30 in. deep ; they are generally calculated to hold about 1 cub. yd. of loose dirt, or from 600 lb. to a ton. It is computed that about 17 cub. ft. of clay, 18 cub. ft. of earth or gravel in the bank, 25 cub. ft. of sand, and 27 cub. ft. of broken earth or gravel when dry will weigh 1 ton. The mine car, which carries the gravel from the breast to the shute, is only half the size of that which carries the gravel from the shute to the outside. These cars carry 400 to 1000 lb.

Cars are generally made of oak, but iron cars are much better, both in durability and economy of running, and weigh less. Mine cars are dumped into a vertical shute with a gate at the bottom, which holds the gravel until an empty dump-car is ready to be filled with it. The tailings, after being discharged in the rivers, are retained there by means of dams to be treated again.

At the Bald Mountain in California, the sluice for washing the deep-lead gravels is 6600 ft. long, 17 in. wide, and 18 in. high, paved with blocks 16 in. square and 4 in. thick. The gold, being coarse, settles in the interstices between the blocks. The usual head of water is 100 to 150 in. The flume is cleaned up in sections twice a month, and a general clean-up is made twice a year, when the worn-out blocks are turned over or replaced. A set of blocks will last about 10 months, but during this time they are turned over. The flume is built on a grade of 6 in. in 12 ft. The main tunnel is 6 ft. 3 in. wide at the bottom, 3 ft. 8 in. wide at the top, and 6 ft. high. It is substantially timbered, and laid with iron tramway-rails. The cars are $4\frac{1}{2}$ ft. long, 2 ft. wide and 2 ft. high, holding 1 cub. yd. of loose dirt. The ordinary car-load is about 16 cub. ft. or 1 ton, and according to the ideas of Californian miners, the pay-dirt should yield 2s. to 4s. per car-load to make drifting profitable ; 1 cub. yd. of rock or hard gravel, &c., will generally occupy about 2 cub. yd. when broken down. "Drifting" is compulsory where there is not sufficient water for hydraulicizing, as well as enough fall and accommodation for the immense masses of tailings caused by that process.

Multifarious devices are employed for conveying the dirt along the drives to the shafts :—throwing it by shovels, carrying it in buckets, wheeling it in hand-barrows, running it in trucks travelling on rails, and either emptying on flats for the dirt to be shovelled to the hoisting apparatus or fitting into cages and raised bodily to the surface. The trucks may be pushed by hand or dragged by ponies, mules, or donkeys, as circumstances dictate. For turning trucks from one drive into another, a plain sheet of iron is preferred to a turn-table. Either water

or steam power may be used for raising the dirt, which is usually tipped into enormous receptacles placed at such a height above the ground as is rendered necessary by the fall of the sluice, which must be sufficiently great to carry away the tailings. Sometimes the water raised from the mine is utilised in the sluices ; but objections to this plan have already been mentioned (p. 230). Where a considerable fall is necessary, it is sometimes obtained by building a tower, to the top of which the cars are run on inclined planes, and there tipped into the sluice. Occasionally these inclined planes are made to radiate to different parts of the working, where the top soil is stripped off as in a shallow placer.

The wash-dirt obtained in driving is either sluiced or puddled : in fact the process becomes identical with that of the shallow placer mining. Sometimes the clearings-up of puddling-machines, pebbles, quartzites, &c., are treated by the ordinary cradle ; the heaps are first sifted, and the fine stuff brought to water and cradled.

When the supply of water, amount of fall, &c., admit of hydraulicing, as described in the next chapter, it is preferred to drifting, as being less costly, and capable of turning out much greater quantities. In this system, shafts are unnecessary, unless it may be now and again for ventilating purposes. The water is introduced at the point where the gravel is being taken out, and the tunnel is made into a sluice, through which all the dirt, auriferous or otherwise, must pass. In one deep lead in California, the gravel is dropped down a shaft 200 ft. deep into the main tunnel. Thence it is washed through 8000 ft. of sluices in the tunnel, and an additional 4000 ft. outside, over the creek-bed and under-currents to the South Yuba, where it is swept down the great cañon into the valley below. Water for power is brought through a sheet-iron main, 15 in. to 7 in. diameter, and branches (7-in.) aggregating 9960 ft. in length, and discharged against "hurdy-gurdy" wheels, 17 to 21 ft. diameter, under pressures varying from 285 to 549 ft. The water-pipe was put together stove-pipe fashion, and gave little trouble by leaking or otherwise.

In a deep lead at You Bet, Nevada county, the cement is overlaid by gravel which is richly auriferous. The cement is being extracted by drifting, and put through sluices. The timbers are taken out as fast as the cement is worked out, and the superincumbent gravel is allowed to fall in, the intention being to treat this gravel by the hydraulic system when the cement is finished. The cement is crushed in a mill of 8 stamps of 850 lb. each, arranged in two mortars. The lift is 12 in. ; 56 drops per minute ; discharge, 5 in. above dies ; sieves made out of No. 18 iron with $\frac{1}{8}$ -in. punched holes ; size of sieves, 48 in. by 16 in. Most of the gold is caught by mercury in the mortars, the rest by copper plates below. The hoisting and pumping are done by the same water-wheel that drives the stamps.

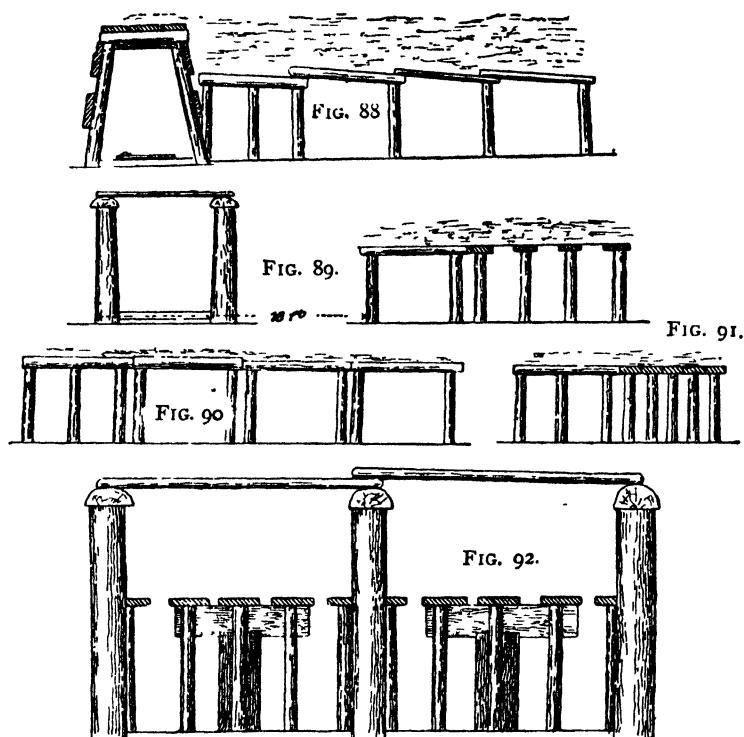
A useful hint for taking water from a river, without the risk incidental to a high dam, consists in digging a cut along the side of the river large enough to contain a box-flume, which will be covered with rock and dirt, leaving the bank of the river in its natural state. Around rocky points, a stone wall is commenced far enough down the banks to get a perfectly solid foundation. This is continued up to the top of the ditch and is made 4 or 5 ft. thick, then an inner wall 2 ft. wide and 2 ft. distant from the outer one is built 4 ft. high from the bottom of the ditch; clay is then tamped between these two walls, so as to make the ditch watertight.

The following description of a system of blocking-out followed in the alluvial mines at Huntly and Bagshot, Bendigo, during the past 7 years is taken from a special report on the subject by H. B. Nicholas, Inspector of Mines at Sandhurst. He says that no accident has occurred from a fall of ground since the system was first introduced in these mines. The main and block drives are timbered, as shown in Figs. 88 and 89. The blocks are commenced at 16 ft. from the main drive, as that distance is considered necessary for the security of the permanent works. In Figs. 88 and 89, the laths overlap, and are placed, in dry ground, the distance apart shown; whilst in very soft or wet ground, each lath stands separately on its own timber, and close together, as shown in Figs. 90 and 91. In each case, the laths and props decrease in size from the drive towards the back of the block, because it is considered desirable to allow the ground at the back of the block to collapse as soon as possible after the "bottom" (wash-dirt) is taken off. The maximum thicknesses of the props and slabs used near to the drives are 5 in. and $1\frac{1}{2}$ in. respectively, and the minimum thicknesses at the back of the blocks are $1\frac{1}{2}$ in. for props and $\frac{3}{4}$ in. for slabs. If the block or roof stands up any time, is swaying, or, to use the miners' term, "talking," the props are knocked from under the back laths, and the block is let down, thus relieving the front laths of the weight, and saving the drive from crushing. This method of blocking is the safest and cheapest way to work alluvial ground. The safest, because each lath stands on its own timber, and there is no shaking of the roof by driving laths, and no fear of knocking the timber out; and the cheapest, because the timber is lighter to handle, cheaper in price, no false sets are used, nor anything cumbersome. If the ground is very soft or heavy, the blocking drives are put in first, and the miners then block back on both sides, and let the ground down in working back. In dry ground, they take the block up on one side of the drive, as shown in Fig. 88, and bring it back on the other, always letting the roof down at the back.

In driving, the sets shown in Figs. 88 and 89 are considered the best; if the ground is very heavy, the cap of the set, over which the laths are

to be driven, is dogged with two strong iron dogs to the back sets which have weight, and a horned false set is used. This is very quickly done if, when the set is made, an auger-hole is bored in each end of the cap; it will then be ready for dogging, and when the set is up the dogs are easily withdrawn. In driving through heavy ground, 4-ft. slabs are generally used, and occasionally slabs of 4 ft. 6 in. In blocking-out, however, 4-ft. 6-in. slabs are always used.

When the alluvial drift in which the mining operations are carried on becomes very heavy, between each set a "dummy" prop and cap are



BLOCKING-OUT DEEP LEADS.

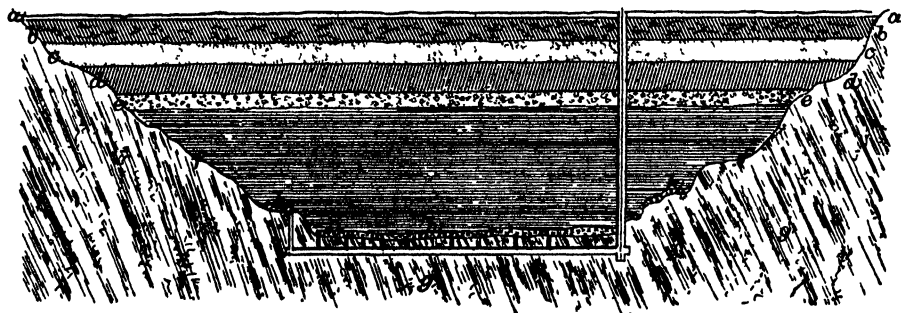
added. The props are 8 to 12 in. diameter, and the caps used are 2 to 3 ft. long by 6 to 8 in. diameter; these are put in to protect the drive, as shown in Fig. 92.

In working very soft ground by this system, and supposing the miner to be in the face or commencing to block out, he takes a lath 6 to 8 in. wide and a prop about 2 in. diameter, each of the required length; he places the prop under the centre of the lath, which he holds with one hand against the face, and, with a light pick in the other hand, he scratches away the drift (sand) in front of the lath until it is in the

full width, then, placing a prop under each end of the lath, he removes the centre prop, and takes out the wash-dirt. One manager of long experience in this division, in describing this method of working soft ground, says, "I have seen miners put in laths by holding the centre in one hand, forcing the lath against the face, and with the fingers of the other hand work out the sand until the lath was in its place, then prop the ends, remove the centre, and take out the wash." He further mentioned, to show the superiority of this system as compared with the old one, that "he had seen a miner working in ordinary ground, using 2-ft. 9-in. props, excavate a block 4 ft. wide by 16 ft. in length in 8 hours, and, at the same time, as his work progressed, he put in 16 laths and 20 props. This was in dry ground, where the laths lapped, and were placed 1 ft. apart." Under the old system of blocking-out, an excavation 4 ft. wide by 8 ft. long is considered a good 8 hours' work.

Where the tunnel would have to be of great length in order to reach the deposit, it may often be found less expensive to sink a shaft directly on to the lead, though that necessitates increased machinery for raising the wash-dirt to bank, and for draining the mine of water, by pumps or other well-known contrivances. Where there is no convenience for a horizontal tunnel, a shaft becomes a necessity. In opening new and unknown leads, it is advisable to bore first, in order to find the *position* of the lead and the most suitable spot for a shaft, then to sink a shaft to ascertain the *character* of the lead, supplementing it with tunnels or drives in different directions if considered desirable. Fig. 93 illustrates the method of sinking and driving as carried out on the South Lead Gold-

FIG. 93.



SINKING AND DRIVING ON DEEP LEADS.

mining Co.'s property, at Forbes, New South Wales, sketched by De Lacy Richards, R.N., from pencil delineations by Phillip Davies, to whom the author is indebted. The reference letters indicate: *a*, surface soil; *b*, clay with limestone clinkers; *c*, very fine drift containing much water;

d, rotten clay ; *e*, gravelly drift ; *f*, hard brown clay ; *g*, slate ; *h*, wash-dirt.

The Table mountain in Tuolumne county, California, already described (see p. 269), was first worked by horizontal tunnels, costing enormous sums of money, on account of their great length. In some cases, these tunnels have been discarded for mining purposes, and allowed simply to serve as drains for the workings, while diagonal tunnels or inclined shafts have been put down on the face of the rim-rock and between it and the basaltic capping. Tunnels or drives radiate from the foot of the incline.

The opinions of a number of practical Australian miners on the subject of the grade of main tunnels and wash-dirt drives may be epitomised as follows :—

Serjeant considers that main levels should be kept as dead as possible, the water running through them towards the shaft having a tendency to keep the discharge end the lowest. Robertson considers that main levels should have only an incline of about 3 or 4 in. per 100 ft., and that a greater incline is a loss in working. The force, he says required to drive empty trucks up an incline of 6 in. per 100 ft. is greater than that required to bring the full trucks down—that is, after they are set in motion. When, however, the quantity of water is very great, 6 in. per 100 ft. is necessary. The same reasons will apply to other drives ; but in the case of wash-dirt drives, the incline is regulated according to the dip or rise of the wash. Smyth mentions that the main levels constructed by the West Ristori Company are driven at a gradient of 4 in. per 100 ft. In the wet or gutter workings the drives rise in accordance with the greater or less fall of the old river bed. Sometimes the incline is 2 ft. in 100 ft., and sometimes as much as 4 ft. in 100 ft. In the Allendale district, the incline averages from 2 ft. 6 in. to 3 ft. in 100 ft. Bryant is of opinion that every main level should have as little incline as possible, say 4 to 6 in. per 100 ft., for the following reasons :—(a) Transport of dirt—More “empties” can be returned, either by men or horses, and there is less wear on the trucks ; (b) In all mines, especially alluvial, where there is water, a quantity of sand and sediment is brought back with the water, and in rapid inclines it silts up the well, and causes more wear to the pit work ; whereas the silt should settle in the drains, where it can be easily attended to, and when the water reaches the shaft it is comparatively clear ; (c) There is more room for storage of water in case of accident to pumps or burst of water, before access to the shaft is stopped by water rising to the cap-piece. Men are always afraid, in rapid inclines, that they will be shut in by the water rising to the top of the drive, near the shaft, before they can escape. In most cases this is provided against by the construction of an escape-drive at a higher level.

Where a large quantity of water is expected in opening up a mine, the main level should be constructed 6 in. higher than usual, to allow for deeper drains on each side of the tramway. In wash-dirt drives, the gradient depends upon circumstances in a great measure, but Bryant always endeavours to do away with inclines, in order to get the maximum of work at the minimum of cost. Langdon states that in his company's mine there is a main level 900 ft. long, with an incline of 4 in. per 100 ft., and there is a quantity of about 10,000 gal. of water per hour running through this level. The truck-road is laid about 1 ft. nearer to one side of the level than the other, and it gives a good space for the water to get away on the wide side. There is also a main reef level going south, which is now (3rd September, 1885) in 600 ft., and the incline is little over 6 in. per 100 ft., with about 3000 gal. of water per hour running through it. Another main reef level going north has an incline of 3 in. only per 100 ft. Langdon would have preferred at least 4 in. per 100 ft. in this drive, but he could not give it, as the drive was not the proper depth under the wash-dirt workings. The truckers find that 4 in. per 100 ft. is the lightest to truck on, especially where main levels are being extended and heavy timber is used, also when large quantities of timber are used in the blocking ground. It is the back loading which tells on the truckers if the incline exceeds 6 in. per 100 ft. Langdon considers that 4 to 6 in. per 100 ft. is the most practical incline for alluvial mines, and in quartz mines the incline should not exceed 4 in. per 100 ft. Henderson is of opinion that 6 in. per 100 ft. is sufficient to carry off the water, while it is very little resistance to the running of trucks on the rails. Nicholas has found that nothing works better than an incline of 8 in. per 100 ft. for main levels. In driving in the direction of the rise in the lead, the incline is often 1 ft. in 100 ft., which is found to answer well, as the drive is kept freer from the accumulation of sand, &c., in consequence of the water washing it down more easily towards the shaft. In driving in the direction of the dip of the lead—and a probability of being too shallow—the incline is 4 in. in 100 ft. only, which is the least incline obtainable where there is water; but if there is much water and sand in the drive, 4 in. per 100 ft. is objectionable, in consequence of the accumulation of sand, sludge, &c. At the No. 2 shaft of the New Australasian mine there is a main drive 1500 ft. from the shaft, with a well-laid road, and an incline of 8 in. per 100 ft., and a lad can easily take four trucks at a time. The full trucks will run easily towards the shaft, and are easily returned when empty. At No. 3 shaft, with the same incline, the company have a horse at work, which takes on an average 20 trucks at a time without difficulty. In driving in alluvial wash one has to be guided greatly by circumstances. As workmen in the wash run only one truck at a time, a manager often rises (when necessary) 2 ft., 3 ft., and even

4 ft. per 100 ft. ; but, where suitable, an incline of 1 ft. per 100 ft. in the alluvial workings is sufficient, as the sand and dirt do not lie on the road, and the trucks run easily each way. Kirk considers 5 to 6 in. per 100 ft. a good incline for main levels in wet ground, but in a mine where the main level has to be extended for a long distance—with even a good flow of water—he prefers 4 in. per 100 ft. Good roads should be kept, and laid with small quartz or tailings between the sleepers, and a good clean drain should be kept on each side, as it is easier for the return load of empty trucks, and for the loads of timber drawn to the face by either men, boys, or horses. An extra dip in the drains should be given as they approach the chamber, so as to convey the water away quicker. In cross drives, 2 to 3 in. per 100 ft. is sufficient, because the water is divided throughout the various drives in a mine. There cannot, however, be laid down any strict rule in mining for main or cross drives, because the gutter is frequently very irregular and uneven in the bottom. Again, a shaft may be sunk, an extensive chamber and water lodgment constructed, and a main drive put in a certain distance, but when the gutter is risen into, it may be found to be deeper than was expected ; consequently, the remainder of the main drive must be constructed dead level to enable the boundary of the claim to be reached. Sometimes even the drive has had to go down a steep gradient, and the water has had to be pumped up. Of course, this can only be done where the water is moderately light, or the shaft must be sunk deeper, with all the heavy expenses attendant upon the construction of chamber, &c., over again. Maughan considers about 6 in. per 100 ft. is the best gradient for main levels, either for short or long distance. There is no rule in wash-dirt drives, as the mine manager has to be guided by circumstances. Lauder states that 9 in. per 100 ft. is a good gradient for main levels where a large quantity of water is constantly flowing. The first 100 ft. from shaft should be perfectly level, to permit the trucks to be stopped gently. An incline of 6 in. in 100 ft. answers very well for intermediate levels. The roads should be raised in proportion where large volumes of water are likely to flow. Thos. Symons (lately of Ellis' Freehold Company, Malmsbury), mentions that no rule can be laid down for an incline for a main level. Water, he says, will flow to the shaft any distance with the least possible fall. It is very difficult to compel men to keep the water-level dead, especially in the main or bottom levels. Where the water is very strong, it carries to the shaft all the slum, and this is pumped up with the water ; but where the flow is light, the slum lodges in the drains, and these must be cleaned out whenever necessary. In the Ironstone Hill mine, when Symons was manager, he had, he states, 3800 ft. of bottom levels, and in that distance he lost 5 ft. 7 in., although great care was taken to keep the levels as dead as possible.

The ground consisted of slate and sandstone. Recently, in the Ellis' Freehold mine, Symons constructed 2570 ft. of bottom levels in slate and sandstone, and lost 6 ft. 2 in. These levels were also put in with much care.

Fig. 94 illustrates the underground workings of the New Australasian Co., at Creswick, as drawn by the mining manager, William Nicolas.

No. 3 shaft was commenced about the beginning of 1880, and the basaltic rock passed through in April of the same year, at a depth of 106 ft. After sinking through alternate layers of clay and drift, the bed-rock was reached in July at a depth of 173 ft. from the surface, and the required depth (175 ft. from the surface) was reached in September 1880. This shaft was also opened at three different levels, viz. 187 ft., 228 ft., and 275 ft. respectively. The main lower level was driven 1400 ft. in a westerly direction before the deep ground was reached, and in opening it up, the wash was found to be poor in quality. After opening up and blocking out about 1200 ft. in length of the deep ground it was found that it could not be worked at a profit; consequently the gutter was abandoned, and the workings from No. 2 to No. 3 shaft not being connected, about 240 ft. of the deep ground yet remains unworked. North of this company, the Red Streak or Australasian Gutter has been proved by other companies with no better results. Finding the deep ground not payable, a prospecting drive was commenced from the intermediate or 228-ft. level, and after driving about 900 ft. in a westerly direction, a payable reef wash was met with, about 350 ft. in width and 50 ft. above the gutter level.

Between the gold-bearing reefs east and the gutter there in an undulating country about 1500 ft. in width, where the gold appears to have found a lodgment before reaching the deep ground. It is to this uneven country that the mining operations have been confined during the last three years.

In the eastern wash, towards the northern boundary, an unusual alluvial formation was met with, and worked with profitable results. This formation consists of a well-formed vertical wash, about 20 ft. in height; it was worked for over 800 ft. in length. The wash referred to was, in some places, in a vertical, at other places in an overhanging, and at others in a slanting underlying position; but even when in a vertical or overhanging position, the wash contained large water-worn quartz boulders, and was as well defined as where worked horizontally. The mode of working was by stoping, as a quartz reef. That this irregularity in alluvial wash is the result of an upheaval can be little doubted. The peculiar folding of the bed-rock, and the continuous and unbroken nature of the wash (of the same character) from the horizontal to the

vertical, are circumstances which confirm this opinion. The upheaval referred to extended along a line of quartz reef, which is situated in a belt of country about 45 ft. in width, running N. 7° E., and composed of black slate, some parts of which are of a tough clayey nature (often termed by miners "leather jacket"), which is heavily charged with minerals and intermixed with quartz veins. Upheavals are frequently met with, particularly in alluvial workings, where the upheaved part extends along the line of cleavage of the bed-rock, and at the same angle of underlie as the cleavage lines. In all cases that have come under observation (excepting the one referred to) there was an absence of wash in the vertical part between the higher and lower washes.

About 500 ft. from the north boundary, a cross-course or clay dyke was met with, which appears to have had the effect of carrying the gold-bearing wash with it in an easterly direction; the cross-course ran nearly east and west, and was traced over 1500 ft. from where it was first met with. The wash found in proximity to the cross-course generally proved to be richer in quality. The gutter varied from 250 ft. to 400 ft. in width, and the bed-rock consisted chiefly of hard blue slate. In the higher formations or reef washes, the bed-rock is composed of a schist formation, and is softer, sometimes approaching to pipeclay in appearance.

Some estimate may be formed of the work done in the mine from the following figures:—Over 15,000 ft. of reef drives have been constructed; about 26 miles of drives in wash; and over 75 acres of ground blocked out. When both shafts were in full work, over 700 tons of wash were put through the puddling machines every 24 hours. The transit of the wash below was effected by horses. The ventilation of the mine has been achieved (very efficiently) by air ducts, for which purpose two have been erected at each shaft. The plant at No. 2 shaft consists of a 22-in. cylinder engine for pumping, two 16-in. cylinder engines for winding and puddling, and cast-iron puddling machines. The drainage of the mine has been performed by a 15-in. plunger and drawlift at each shaft. The plant at No. 3 shaft consists of an 18-in. cylinder engine for pumping, two 16-in. cylinder engines for winding and puddling, and three cast-iron puddling machines.

A very simple yet effective process for raising the wash-dirt is adopted where sluicing is inadmissible. The shaft is divided in the centre. In one division a small tank is filled with water from the race, and when ready is lowered into the shaft; by its weight it brings up in the other division of the shaft a truck of wash-dirt, which is then removed to a paddock for washing, and the truck is replaced empty in the shaft. The water is released from the tank by means of a valve, and runs down the tailrace, and the wash-dirt truck, by its own weight, brings up the empty water tank.

Ventilation.—A few of the simplest modes of ventilating are worthy of passing notice. One of the earliest plans adopted for ventilating a shaft was by means of a calico windsail, such as is used on ships for taking air down into the hold, suspended from a pole, the wings kept open by means of cords fastened to pegs, and the pipe hanging down the shaft. This apparatus is not capable of forcing air along the drives. The “fanner” consists of a spindle furnished with wings, which is made to revolve by an endless cord passing round the wheel on the spindle and another which is turned by hand, water, or steam power. The piping may be of calico, tin, or galvanised iron. A third plan consists in dividing the shaft by a partition running down the centre and parting the main drive in a similar manner. In order to clear the drives of foul air, a quantity of water is thrown down one compartment of the shaft, so as to displace the air and force it through the drive and up the other compartment of the shaft. Other plans which have been used are :—carrying air-drives in the same direction as the main drives at a higher level, and constructing openings at various points from one drive to the other ; building a furnace underground in connection with the air-drives ; erecting an air-stack over an air-shaft, connected with one of the compartments of the main shaft. Besides these, there are the air-duct, and air-engines of all descriptions.

Ventilation of the tunnels requires attention, but is rarely difficult. The workings are sometimes connected with the surface by a tunnel or shaft, or both, independent of the bed-rock of the tunnel. In such a case there would be a temporary tunnel in the drifting ground and a permanent one in the bed-rock. The tunnel then becomes an air-way, and is connected with all the cross drifts on one side of the ground which is being drifted, and not through the middle of it. This gangway is always well timbered, and the ground in its vicinity must not be disturbed while the mine is being worked, as any caving which would close it might result in serious accidents and loss of life. It is used for ventilation, for access to the mine, and for traffic. When there is not a sufficient difference of elevation of temperature between the two tunnels to cause the air to circulate, a shaft is made connecting the air-gangways with the surface, and an artificial draft is induced by lighting a fire. When there is only one opening in the mine, ventilation must be secured by fans, run usually by water power.

Apparatus.—The gold-extracting apparatus generally used where the deep leads are worked by drifting consists of puddling-machines, sluices, &c. At one large Victorian works, 2 puddling-machines each 16 ft. in diameter, and a sluice-box 60 ft. long and 10½ in. wide, were used, the average quantity of wash-dirt put through each machine per 24 hours being 300 trucks or 60 tons. At another lead on the Ballarat gold-field ant consisted of 5 iron puddling-machines, 2 sludge-machines

2 buddles, and 2 sluices. The sluices were for separating the gold from the dirt after it had been through the puddling-machines, and the sludge-machines and buddles were used for saving the fine gold carried off in the sludge from the puddling-machines. The sides and bottoms of the puddling-machines are sometimes covered with $\frac{1}{8}$ -in. sheet-iron. In a third instance, are 4 cast-iron puddling-machines, fixed so as to form a square; each machine is 16 ft. 6 in. in diameter and 2 ft. 6 in. deep, and worked by Hunt and Opie's patent chain and pulleys. Above the machines is a stage connected with the brace, which is about 40 ft. above the surface, and upon this stage trucks are run and the dirt emptied into the machines. In the bottom of each machine are two cast-iron doors, about 1 ft. 6 in. by 2 ft., hung so as to open below, for the purpose of cleaning them out. These doors are fastened with 4 screws, and when the machines have been filled and the dirt sufficiently puddled, the doors are opened from below, a barrow and scrape are attached to the machinery, and by means of these the dirt is cleaned out of the machine through the doors in about 30 minutes. Below the machines is the sluice-house, where are 4 sluice-heads, one under each machine; these lead into the centre or main sluice. The dirt, when puddled, falls through the holes or doors in the bottoms of the machines close to the sluice heads, so that the men can conveniently feed the sluice. In the centre sluice is a fork worked by machinery, for the purpose of separating the dirt. The four machines are tended and cleaned out by 2 men in each shift, and all the dirt is sluiced during daytime by 2 men and 1 boy. The stones taken from the machines, as the dirt is puddled, are passed through the centres of the machines, so that no trucks or barrows are required on the machine floor. The two iron sludge-machines are about 14 ft. in diameter and 2 ft. deep; the sludge as it leaves the puddling-machines passes into them, and the sand taken from these machines is passed through a sluice and over a blanket surface; the stuff washed from the blanket is put through the amalgamating barrel.

The mill of the Paragon Company, in Placer County, California, has 10 stamps, of 850 lb. each, with steel shoes and dies, dropping 90 times per minute, with a fall of 10 in. The capacity of the stamps is 40 tons per 24 hours. The screens are sheet-iron, with $\frac{3}{8}$ in. round holes. The coarser gravel and boulders are not crushed, but only the finer and softer gravel. The greater part of the yield of this mine comes from the stamps. From 60 to 70 per cent. of all the gold saved is caught by amalgamation in the battery, 20 to 25 per cent. on aprons, and the remainder by a concentrator and in 150 ft. of sluices. A car-load contains, on an average, 28 cub. ft., and weighs about $1\frac{1}{2}$ ton. The yield per ton was \$7.05. The gravel is dumped from the track of the tunnel to the mill platform, where large rocks are picked out, and carried to the dump head, which is occasionally sluiced and yields some gold.

All the other material is fed by hand to the mortars, which have a single discharge. The pulp flows over a table 8 ft. long, provided with grooves or riffles charged with quicksilver. It then passes through rubbers to the concentrator where a small portion of the gold is saved.

At the Centennial Mine the gravel is hoisted through an inclined slope 200 ft. in length and dumped into a bin, from which it passes through an automatic feeder into a revolving cylinder of boiler iron, provided in its interior with railroad rails. The cylinder is 20 ft. long and 3 ft. in diameter. It works like a Brückner furnace. The gravel is thrown in, and in the length of the cylinder is supposed to fall about 13 times. The result is that it is broken and cleaned. No stamp mill is used. At the lower end of the cylinder there is a set of bars which are about 3 ft. long and $1\frac{1}{4}$ in. apart, which allows the fine portion to pass through. This falls into the feed-box of the stamp-mill, while the large stones fall into a car to be thrown away.

Working Results.—The method of stating the yield varies. In some places, the standard is the car-load, calculated at 16 – 20 cub. ft. = 1 ton ; but, as the size of the car depends upon the size of the tunnel, the car-load, without giving dimensions, cannot be a standard of value. About 9 cub. ft. of ground in place, when broken, will fill a box of 12 cub. ft. capacity. An estimate, based upon cubing the feet, without explanation, would be erroneous. Another way of estimating the yield is per ft. of channel. In some places the value is expressed at per acre or at per superficial yard of channel-ground, but the height of the ground is not uniform. In some places it is 3, and in others 4 or 5 ft. The only way to definitely express the value is by the cub. yd., but this method is not in general use, so that it is not easy to get a standard of value for comparison.

To pay, gravel should contain, under ordinary circumstances, 2s. to 4s. worth of gold per car-load of 16 cub. ft. The minimum value of ground to be worked should thus be 2s. to 4s. per cub. yd. of gravel extracted.

The following figures concerning the yields of gold from various deep leads may be interesting, though affording no guide to the possible returns of any new field of operations. One field of 20 acres produced 16,440 oz. of gold ; the average at another diggings was 2 dwt. 22 gr. per ton ; at a third, 1 to $1\frac{1}{2}$ dwt. per load ; at a fourth, 5 dwt. per ton ; at a fifth, 6 dwt. per ton ; at a sixth, 15 dwt. per load. In a seventh instance, one mile of lead yielded 170,000 oz., equal to 32 oz. per lincal ft. These figures all refer to Victoria, and are on the authority of Brough Smyth. In California, 50 c. (say 2s.) per cub. yd. seems to be the average value of the wash-dirt in the deep leads ; but Raymond mentions four Companies which made respectively \$10·80, \$8·10 \$9·45, and \$7·38 per sq. yd., which gives an average of 95 c. (3s. 11½d.) per sq. ft. of ground $4\frac{1}{2}$ ft. high, or \$5·70 (23s. 9d.) per cub. yd. of gravel.

WORKING DETAILS OF TUNNEL CLAIMS NEAR DUGTOWN, CALAVERAS COUNTY, CALIFORNIA *

	Hammer chmidt Hensel & Co	Barney Hurle & Co	Bully Company	Buckeye Company	Detrich & Co
Inches of water used in sluice washing	25 to 30	30	30	30	30
Inches of water used in hydraulic washing	20	6 to 8 months	6 to 8 months	6 to 8 months	6 to 8 months
Height of fall	100 feet	\$4	\$4	\$4	\$4
Supply of water lasting in the year	6 to 7 months	One of 105 feet	4	4	10
Cost of water (sluicing) per day	\$4	One of 200 one of 400, and one of 300 feet	700 feet	1000 feet	1200 feet
Cost of water (hydraulic) per week	\$20	4 1/2 to 5 1/2 feet	4 feet		
Air shafts	1500 feet	In 1862	1870	1870	18 0
Length of tunnel at present	6 feet	5600 superficial sq yd	3000		
Height of drift (all pay dirt)	In 1865	7500 sq yd	1000 feet		
When this claim commenced	9166 superficial sq yd	600 feet	Over one half		
Area of ground drifted out and worked	18,332	15 to 20 sq yd	2000 sq yd		
Number of cubic yards drifted out	400 feet	Over one half	\$700		
Length of ground unworked	20 to 25 car loads, about 25 sq yd	Not washed			
Average of cement gravel extracted per day	Over one half	\$4			
Quantity of refuse left in the slope	2200 sq yd	Not washed out			
Quantity of gravel cement washed during last year and extracted	\$4 to \$5 per sq yd	\$30 000			
Yield of cement gravel washed	Over \$8000	Both coarse and fine			
Total yield last year	\$70 000	\$2 50			
Total yield since commencement of this claim	Small flat particles	4			
Gold, description	\$2 50	4			
Wages paid to underground drifters	4	4			
Number of men hired	8	Red soil, quartz, gravel, slate, granite, sand—all cemented			
Number of men working	Red soil, small and large white and blue quartz, boulders, pieces of dark blue slate, quartz, and mica slate, cemented granite gravel				
Composition of deposit	Dark blue slate, full of cubical pyrites.				
Bed rock	E N E	Dark blue slate, veins of quartz	Blue slate	Blue slate	Blue slate
Direction of tunnel and course of deposit channel		E	S E	S E	S E.

* Mining Commissioners Report, 1872

In one of the Government Surveyor's Reports on New South Wales, the dimensions and prospects of a particular lead are succinctly placed before the reader. The width of the lead is about 200 ft. ; thickness of wash-dirt, 3 ft. ; yield of gold, low—5 dwt. per ton ; every 100 ft. run would produce 60,000 cub. ft. of wash-dirt, or 2500 loads of 24 cub. ft. to the load, which, at 5 dwt., would yield 625 oz. or about 2500*l.* per 100 ft. This authority says that deep leads can be worked (in New South Wales) at a cost of about 10*s.* per load, including puddling the wash-dirt by steam-power ; 30 to 40 men could work on a lead 200 ft. wide, and 30 men should produce 360 loads per week, yielding 180*l.* clear profit per week, after paying 10*s.* per ton for working expenses. Hundreds of thousands of tons of wash-dirt have been worked at Ballarat, in Victoria, for considerably less than 10*s.* per ton.

CHAPTER V.

HYDRAULICING: OR THE USE OF WATER UNDER GREAT PRESSURE
FOR REMOVING AND WASHING GRAVELS.

Origin.—In the last chapter were discussed the features of deep leads and the means adopted for extracting their gold by the aid of hand-labour and moderate supplies of water. But hundreds of square miles of auriferous gravels exist whose gold is so fine and so widely distributed that its extraction could not by these means be made to pay. The disintegrating and removing power of water is patent in every-day life: it has been the great agent in forming the deep and shallow placers; and without it the modern miner would be unable to separate the gold from the baser bodies with which it is mixed up in the leads, and in quartz veins. A sharp-witted American, foreseeing that the effects of an ordinary stream of water, acting by its own gravitation as in a gentle current in a sluice, would be many times magnified by causing the water to fall from a considerable height, constructed an apparatus which was the initiation of hydraulicizing. He cut a small ditch in the side of a hill, and built a flume from it to a point 40 ft. above the place where he was working. Here the water entered a common barrel, whose bottom was filled with a cowhide hose 6 in. in diameter, ending in a tin tube 4 ft. long tapering to a 1-in. nozzle. The advantages of the plan were quickly realised, canvas* superseded cowhide, to be replaced in its turn by iron piping, and from the rude attempts of an obscure American miner a new science has grown, termed "hydraulicizing" (hydraulicizing was the original spelling) or "piping" in America, and "ground sluicing" in New Zealand.

Advantages.—It has already been shown that, as a general rule, with very few exceptions, the gravels of a lead are very poorly auriferous throughout the upper strata, and that the gold is found almost entirely in a layer of a few inches or a few feet on the bed-rock. Where the gravels have to be removed by hand for treatment in the sluices and other apparatus, it becomes important to confine the labour to the rich bottom stratum. In hydraulicizing, on the other hand, the whole depth of the deposit down to the bed-rock must be taken away, no matter

* Canvas hose and brass nozzles are still in use in New Zealand in localities where a moderate pressure only is needed.

how great the depth may be. This has to be performed by the force of a powerful jet of water, which same water must effect the disintegration of the gravels in the sluices. Consequently the first and greatest *desideratum* is a large and constant supply of water. The supply of water once ensured, the process of the washing is conducted with astonishing economy. Laur,* who was sent to California to report on the gold-mines there to the French Government, estimated that the cost of treating 1 cub. yd. of auriferous gravel by the various methods, would be about as follows :—

					£	s.	d.
With the pan, about	..	..	..	..	4	0	0
„ rocker, about	..	..	..	..	1	0	0
„ long-tom, about	..	..	..	..	0	4	0
„ hydraulic process, about	..	..	..	..	0	0	2½

These prices are assuming the rate of wages to be about 16s. a day. The figure for the hydraulic process includes the cost of water.

Essential conditions.—Before commencing a hydraulic washing, the first thing to be done is to ascertain the total depth of the deposit from the surface to the bed-rock, and to test the richness of the strata throughout, as this will be found to vary from ½d. to 4s. or 5s. per cub. yd.

As the whole deposit must be removed and treated, it is essential to study these points with great care, in order to form an estimate of the probable returns, before expending large sums of money. For these purposes, mere boring, though useful as a preliminary guide, is not sufficient. Shafts must be sunk at such points as will indicate the depth and extent of the deposit, the richness of the various strata, and the character of the bottom-rock, in order to be sure that the quantity and quality of the gravel are such as to warrant the undertaking, which will require the expenditure of thousands of pounds without giving any return perhaps for several years. Besides these precautions, accurate topographical surveys must be made, with the view of ascertaining (1) what head or fall of water can be got, and (2) what outlet can be secured for the tailings. These two considerations have equally important bearings upon the success of a hydraulic washing, and, unless they receive proper attention, disappointment and loss are sure to follow.

The first point is seldom suffered to escape due attention ; but nothing is more common than to see works stopped, and the expenditure of enormous sums of money rendered fruitless, by a neglect to provide sufficient accommodation for the gigantic masses of earth which are removed. The outlay entailed by a hydraulic mine cannot be justified, except the work be permanent, and sufficient gravel be at hand to last

* Laur's estimate, as quoted by the Commissioners for Victoria at the Philadelphia Exhibition, is,—Pan, 3d. ; rocker, 16s. ; long-tom, 4s. ; sluice, 1s. 6d. ; hydraulicing, 3d.



GROUND-SLUCING AT DEVIL'S CREEK, REEFTON, NEW ZEALAND

for several years' operations, consequently "dump" fall or outlet must be secured for the millions of cubic yards of earth displaced, and it must not be forgotten that, in a loosened condition, it will probably occupy twice the space which it filled *in situ*. Numbers of valuable undertakings are suspended through careless inattention to this precaution, and many more would be but for the spring freshets, which help so much to remove the accumulated deposits. Frequently it happens that sufficient fall cannot be obtained from the bed-rock—and of course the drainage must take place from the bed-rock or lowest part of the working—without driving tunnels (it may be several miles in length) under the bed-rock and even under valleys and through hills, in order to reach a river-valley lying at a level low enough to receive the drainage, and of a capacity adequate to the accommodation of the tailings. The primary cost of opening a hydraulic working, without taking into consideration any expenses for water, will often reach 10,000*l.*, and is sometimes double that amount. It is not too much to say that, other conditions being equal, this sum is risked upon the judgment and care with which the mine has been laid out. The location and inclination of the tunnels and sluices have almost a greater bearing upon the success or failure of a hydraulic mine than has the proportion of gold contained in the gravels, for while 2*d.* worth of gold per cub. yd. has given good returns under favourable conditions, scarcely any amount within the range of what past experience has made known could pay for elementary blunders in the laying out of the workings.

Water supply.—Primarily then, the success of working deep placers is dependent less upon the yield of gold than upon the constancy and amount of the water supply. In America, where hydraulicing is principally developed, this has given rise to a number of so-called Ditch Companies. Originally the corporations of miners collected water solely for use on their own diggings; then they found willing purchasers for any excess they might have beyond their own particular needs; and soon it became evident that to be purveyors of water was more profitable and less risky than to consume it in working gold-diggings. Thus many companies which started as gold companies have in time become ditch companies, and look upon the gold-getting as a secondary consideration. As an illustration of this, mention may be made of a company which expended about 9000*l.* in preliminary operations (before the bed-rock tunnel had been driven), and took out about 10,800*l.* in gold, leaving a profit of 1800*l.* Had it sold the water instead at about 8*d.* per in. per 24 hours, it would have reaped a profit of about 8000*l.*

Division of the subject.—There are thus two distinct phases in hydraulicing:—(1) Collecting and conducting the water to the diggings; (2) Applying the water to the extraction of the gold. When the under-

taking is of a limited description, these operations may be conveniently combined; but in the Western States of America, the enterprises are generally of such magnitude, and the water-supply question is of such weight, that there is a growing disposition to keep the two branches of the industry distinct, the ditch company selling its water for every purpose to which it may be applied, and the gold company buying its water from the ditch company, and confining its attention to the mining proper. The water is sold generally at prices varying between 5*d.* and 10*d.* (10 to 20*c.*) per "miners' inch" per diem, the day being variously estimated from 10 to 12 hours.

Miners' "inch."—It may be well here to state what a "miners' inch" is. It is an arbitrary measure of the quantity of water which will flow through a given space in a given time, adopted in the early days of American gold-mining, and established by the law of each miners' camp, without any attempt at a universal scale. Thus there are scarcely two localities where the miners' inch has the same signification, the size and shape of the outlet and the manner of discharging the water varying constantly.

The most common way of estimating the "inch" is the amount of water which will pass through an opening 1 in. square in a plank 2 in. thick, with a pressure of 6 in. above the opening or 7 in. over the centre. The thickness of the plank is sometimes 3 in. The lower front end of the discharge is usually chamfered. Raymond says that with an aperture of 1 sq. in. and a pressure of 6 in., the "inch" will equal 94·7 cub. ft. per hour. In other localities, the pressure used is 10 in., making 109·1 cub. ft. per hour. The average Californian miners' "inch" he puts at 100 cub. ft. per hour, or 1000 cub. ft. per day of 10 hours. He adds the following table of the standard miners' inch :—

Pressure from Surface to Top or Middle of Orifice.	Miners' Inch.	Cub. Ft. (each 6·23 Gal.)				Authority.
		Per Second.	Per Minute.	Per Hour.	Per 24 Hours	
in.						
6	1	0·039	2·34	140	3,360	Hittell.
"	1	0·026	1·57	94·7	2,274	Carpenter.
"	38	1·000	60·00	3600·0	86,400	"
"	1000	26½	1580	94,700	2,274,600	"
10	1	0·03	1·8	109·1	2,618	"
6 to 10	1	0·027	1·6	100	2,400	} Standard experimental miners' inch.
"	10	0·27	16	1,000	24,000	
"	100	2·7	166	10,000	240,000	
"	1000	27	1666	100,000	2,400,000	

The Milton Co. reckon a flow through an aperture 12 in. wide and 12½ in. high, when the water stands 6 in. above the top of the opening, as 200 "inches."

Raymond observes that the "usual acceptance of the miners' inch is

that given by Hittell," and he quotes the following formula from Haswell for making the calculation:—

$$\frac{1}{2} b \sqrt{2g} (h' \sqrt{h' - h} \sqrt{h}) C = V;$$

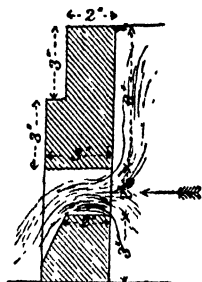
b being the breadth, h' the distance from the sill to the surface, and h the distance from the top of the opening to the surface in feet, while C is the coefficient of discharge assumed at 0.750, and V the volume in cub. ft. per second; thus—

$$\frac{1}{2} \times \frac{1}{12} \sqrt{2g} (\frac{1}{12} \sqrt{\frac{1}{12} - \frac{9}{12}} \sqrt{\frac{9}{12}}) \times 0.750 = 0.031 \text{ nearly.}$$

He thinks the coefficient of discharge is perhaps a little too high.

Egleston states that the "quantity of water which will flow from an opening 1 in. square will be 93 lb. per minute. The opening, however, is never made 1 in. square, but is always longer or higher, which will necessarily increase the quantity of water which will issue from each sq. in. Thus the quantity of water which is given by a single sq. in. opening will be 93 lb. per minute, and from an opening 1 in. $\times$ 2 in. will be 196½ lb., and with 1 in. $\times$ 100 in. each inch will pass 111 lb. Sometimes the opening is 2 in. high. At North Bloomfield, it is 48 in. long $\times$ 2 in. high, with a pressure of 9 in. above the opening. Each square inch will thus furnish 4252 cub. ft. per 24 hours. At Eureka, the opening is the same, 48 in. $\times$ 2 in., with 6 in. pressure above the top of the opening, so that each square inch will furnish 3240 cub. ft. per 24 hours. . . . The La Grange inch is equal to 2.159 cub. ft. To determine the value of the North Bloomfield 'inch,' the water was made to discharge through a 3-in. plank. The bottom of the opening was above the bottom of the tank holding the water (Fig. 95), and was chamfered off 1 in. from the outside, so that the outside opening was 4 in. high. The value of this 'inch' is given in" the following table, which "shows, according to Bowic, the variations in some of the 'inches' and their supply:—

FIG. 95.



MINERS' INCH.

Name of Mine.	Height of Opening.	Length of Opening.	Pressure over Centre of Opening.	Quantity discharged by 1 sq. in. per minute.	Quantity discharged in 24 hours.
Smartsville	in. 4*	in. †	in. 9	cub. ft. 1.76	cub. ft. 2534.4
Park Mining Co. ..	..	..	..	1.39	..
North Bloomfield ..	2	50	7	..	..
Eureka	2	48	7	..	..

* The bottom of the aperture is on a level with the bottom of the box.

† An opening 250 in. long $\times$ 4 in. wide will discharge 1000 Smartsville 'inches.'
The day is usually reckoned as 24 hours.

	H. Smith, 1874. cub. ft.	A. J. Bowie, 1876. cub. ft.
One miners' in. discharges in 1 second ..	0'2624 ..	0'2409
" " 1 minute ..	1'5744 ..	1'4994
" " 1 hour ..	94'4640 ..	89'9640
" " 24 hours ..	2267'1360 ..	2159'1460
Ratio of actual to theoretical discharge ..	61'60 per cent.	59'05 per cent."

On the same subject, Raymond remarks in another report the discrepancies among different companies: thus—

	Height of Aperture	Pressure.	
Eureka Co. ..	2 in. ..	6 in. ..	The amount delivered by them through 20 in. x 2 in. is considered 40 in.
Excelsior ..	..	10 ..	} Measured from centre of orifice.
Sears ..	..	10 ..	
Mokelumne ..	..	4 ..	
Campo Seco ..	..	4 ..	
Phoenix ..	3 in. ..	4 ..	Over the orifice.
Gold Hill ..	2 ..	4 ..	An inch wide.
Another ..	3 ..	none ..	An inch wide.

"At Smartsville, water is sold with a head of 9 in., with a 14-in. opening 125 in. long, giving 11'8 per cent. for an 'inch' more than is usually given. The quantity discharged through an opening 4 in. deep, with a 9-in. head over the middle of the opening, with the coefficient of discharge = 0'0615 is 106'6 cub. ft. per hour, or 1'7767 cub. ft. per minute. A 'head of water' is 500 in. daily for 10 hours, and is the quantity required for a first-class hydraulic operation."

Egleston gives the theoretical horse-power of the miners' "inch" as follows:—

Heads in feet ..	..	100 30	90 20	80 15	70 10	60 5	50 3	40 1
Inches to horse-power ..	..	3'25 10'8	3'61 16'2	4'06 21'6	4'64 32'5	5'41 65	6'50 108	8'12 325

Raymond estimates that "with a moderate ditch delivery of 4000 in., or 5 heads at 800 in., the work done may be 1 cub. chain or 10,000 cub. yd. per day, or in a 10 days' run, an acre 1 chain deep or 100,000 cub. yd. Taking an average of 2000 cub. yd. per day of 10 hours, moved by 300 in. of water, 5 days would move a cub. chain or 10,000 cub. yd.; 800 in. at 100 ft. head working 10 hours = 800 10-ft. cubes of water = 800,000 cub. ft., weighing 24,880 tons, without adding the pressure arising from the head employed. This will move through ordinary sluices, at a grade of 8 to 12 in. per box, 3000 cub. yd. of loosened gravel, or 2000 cub. yd. of ordinary uncemented bank gravel, say an average of 2500 cub. yd., weighing 8300 tons or $\frac{24880}{8347} = \frac{3}{8}$ of the weight of the water employed. Reckoned by 'inches,' the amount of gravel moved = 3 times as many cub. yd. as there are miners' 'inches' used.

"A cub. ft. of water at 62° F. weighs 63·321 lb.; 1000 cub. ft. = 31·160 tons.

Clay	= 120	lb. per cub. ft. =	4800 per yd.	Sp. gr. =	1·92 (Water 1·00)
Sand, dry	= 88·6	"	=	"	= 1·42
" wet	= 118	"	=	"	= 1·90
Trap rock	= 170	"	= 4500	"	= 2·72
Basalt	= 187·3	"	= 5060	"	= 3·00
Quartz	= 165	"	= 4450	"	= 2·65
Shale	= 162	"	= 4370	"	= 2·60
Slate (clay)	= 180	"	= 4800	"	= 2·90
Decomposed shale } estimated	= 100	"	= 2700	"	= 1·80

"The breadth, depth, and velocity of a stream in feet per minute as travelled by a chip, may be estimated by the eye. The sectional area being reduced to sq. ft. and decimals, we have multiple $\times 60$ = cub. ft. per hour; divided by 100 = miners' 'inches.' Or, observe 6 seconds, and the distance $\times$ area $\times 6$ = miners' 'inches.'

"The miner's 'pan' contains about 400 cub. in. of loose dirt."

The results of some interesting experiments to test the relative carrying powers of one 20-in. sluice and two 10-in. sluices, communicated by F. L. Vinton to the 'Engineering and Mining Journal,' of New York, may be summarised in the following tables:—

First Experiment—Sluice Level.

Area of discharge	0·139 sq. ft.	Actual discharge	0·276 cub. ft.
Velocity	1·985 ft.	Theoretical discharge by	
Mean depth	0·13 "	formula $Q = 5·6 a \frac{H}{\sqrt{H \times h}}$	0·269 "

Second Experiment—Sluice Level.

Area of discharge	0·347 sq. ft.	Actual discharge	0·702 cub. ft.
Velocity	2·02 ft.	Theoretical discharge	0·716 "
Mean depth	0·23 "		

First Experiment on Graded Sluices; grade $\frac{1}{8}$ in. to box of 12 ft. = $\frac{1}{81}$ = 0·0017.

Area of discharge	0·295 sq. ft.	Actual discharge	0·657 cub. ft.
Velocity	2·22 ft.	Theoretical grade by	
Perimeter	2·02 "	formula $\sin. = \frac{P}{A} \sqrt{\frac{1}{1000}}$	0·0033 ft.
Grade	0·0017 ft.		
Area of discharge	0·147 sq. ft.	Grade	0·0017 ft.
Velocity	1·9 ft.	Actual discharge	0·279 cub. ft.
Perimeter	1·18 "	Theoretical grade	0·0028 "

Double sluice carries 0·177 more than twice the single on this grade.

Second Experiment on Grade; grade $\frac{1}{4}$ in. to 12 ft. = 0.0035.

Area of discharge	0.295 sq. ft.	Grade	0.0035 ft.
Velocity	2.271 ft.	Actual discharge	0.671 cub. ft.
Perimeter	2.02 "	Theoretical grade	0.0035 "
Area of discharge	0.147 sq. ft.	Grade	0.0035 ft.
Velocity	2.00 ft.	Actual discharge	0.287 cub. ft.
Perimeter	1.18 "	Theoretical grade	0.0032 "

Discharge of double sluice is 0.168 more than two single on this grade.

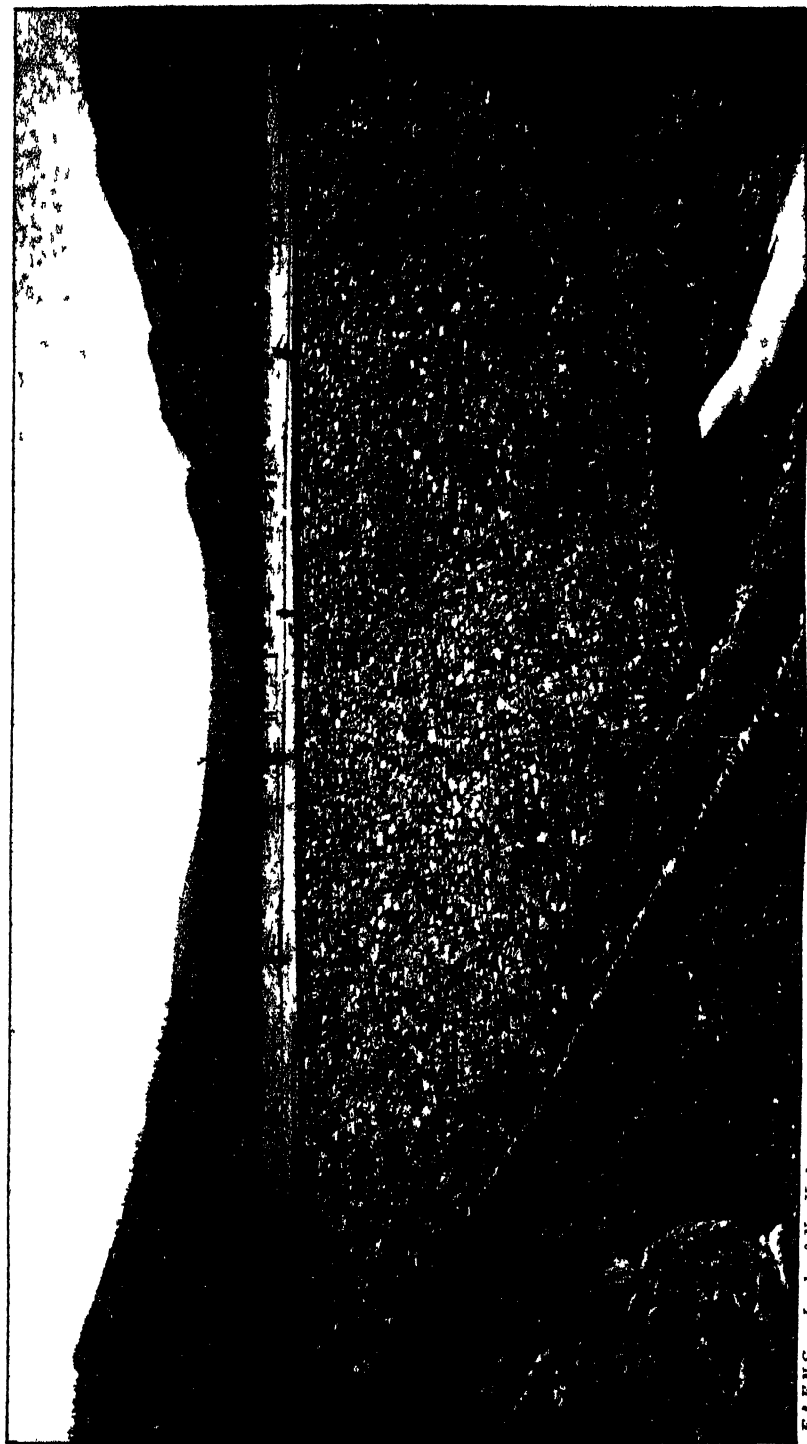
Third Experiment on Grade; grade $\frac{3}{8}$ in. to 12 ft. = 0.0052.

Area of discharge	0.295 sq. ft.	Grade	0.0052 ft.
Velocity	2.8 ft.	Actual discharge	0.827 cub. ft.
Perimeter	2.02 "	Theoretical grade	0.0053 "
Area of discharge	0.147 sq. ft.	Grade	0.0052 ft.
Velocity	2.21 ft.	Actual discharge	0.325 cub. ft.
Perimeter	1.18 "	Theoretical grade	0.0035 "

Discharge of double sluice is 0.272 more than two single on this grade.

Securing Water Supply.—The greatest engineering skill is required in hydraulicing. In choosing the site and line for a "ditch," the first consideration must be that it shall have a full supply of water all the year round. To secure this object, no outlay should be considered too great. The summer supply is more important than it seems at first sight. The days are then long, and in cold regions the workmen are less exposed to the inclemencies of the weather. The heat of the sun renders the water almost tepid, and on this account the mercury is much more free and energetic in its action, giving a proportionately larger yield of gold. The natural supplies of water vary with the seasons: at one time there may be a heavy flood, at another a long drought; for this reason, dams and reservoirs have to be built, to catch and deliver the water as it may be needed.

Dams and Reservoirs.—The object of these constructions is to take in not only the running water supply, but all the water from rains, floods, and snows melting on the hills, and to store it in the spring and summer months for use during the dry season. Dams are usually built where the opening is narrow and rock-walled, the material used varying with the nature of the local supply. Sometimes cut granite blocks are keyed together for the face, the backing being composed of puddled clay and earth. Wooden dams are often made by cutting down the neighbouring trees, running them from shore to shore, 6 to 8 ft. apart for a width of 40 to 50 ft., then placing other trees at right angles to these, about the same distance apart, building up a sort of crib. Where the trees cross, they are fastened together with bolts, and the intervals are filled in with rubble. The trees lying parallel with the stream are usually put in with part of their branches on, and heading up stream. It should, if possible, be arranged for the outlet to be in solid rock.



E & F H Speer London & New York

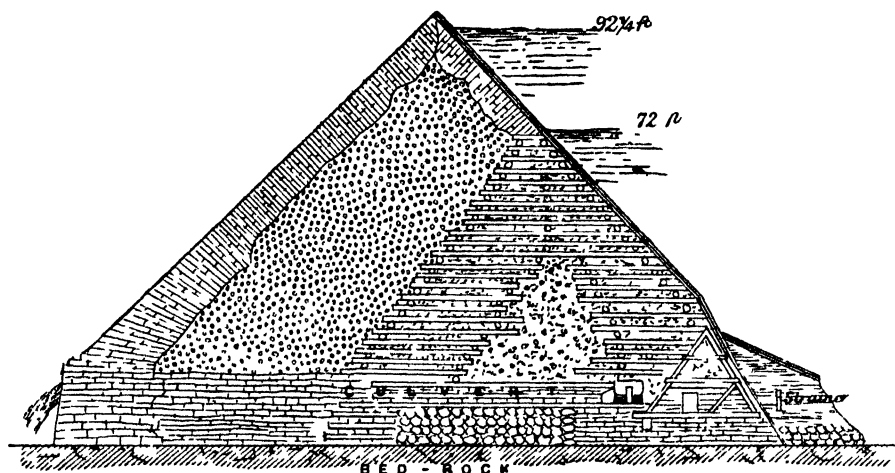
BOWMAN DAM

OF THE NORTH BLOOMFIELD HYDRAULIC MINE, NEVADA COUNTY, CAL

NK PHOTO SPRAGUE & CO LONDON

The following details concerning one of the largest dams in America (the Bowman) possess great interest. The dam was first built to a height of 72 ft., as indicated in Fig. 96, consisting of a timber crib of unhewn logs of cedar and tamarack woods, notched and securely bolted together, and filled in with loose rubble. A water-tight lining was formed by spiking pine planking to the water face. Subsequently the height was increased to 96½ ft. above datum line (100 ft. total height); but at 85 ft.,

FIG 96.



DAM FOR HYDRAULIC RESERVOIR.

it was thought advisable to let a stream of water, 50 to 75 cub. ft. per second, flow over the summit and percolate through the stone embankment, so as to settle the structure before adding the top courses.

The increased height was effected by filling in a stone embankment on the lower side of the old structure, faced with heavy walls of dry rubble of large size. The lower face wall is 15 to 18 ft. thick at bottom, diminishing to 6 to 8 ft. at top, the stones varying from $\frac{3}{4}$ to $4\frac{1}{2}$ tons each in weight, with many equally large stones in the backing. The under portion is $17\frac{1}{2}$ ft. high, with a 15 per cent. batter, built of heavy stones in horizontal layers, and the face stone tied to the backing with iron dowels. The upper part of the wall is sloped at 45° , and the face stone is embedded at $22\frac{1}{2}^\circ$, thus dividing the angle between a horizontal bed and a bed at right angles to the face, no attempt at ranging being made in this portion. Above the 68-ft. line ribs of flattened cedar 8 in. thick are built into the upper face wall, and tied to it by iron rods 5 ft. long $\times \frac{3}{4}$ in. diameter, and to these ribs is firmly spiked a planking of heart sugar pine 3 in. thick $\times$ 8 in. wide, with planed edges fitted with an outgate like ships' planking. The

planking was put on almost thoroughly seasoned, and would swell sufficiently to make the face practically water-tight without caulking or battening the joints. The openings at the joints made by the outgate suck in small particles of vegetable matter, which in a great measure take the place of caulking.

At the bottom, the plank is fitted neatly to firm bed-rock, and caulked with pine wedges. There are three thicknesses of plank (total 9 in.) on the lowest 25 ft., two thicknesses (6 in.) on the next 35 ft., and one thickness on the uppermost 36 ft. Experience justifies the belief that this will remain sound for at least 20 years, when it can easily be renewed. The dam is traversed by a culvert, through which the water is drawn from the reservoir. This is built of heavy dry rubble foundation and walls, and covered with granite slabs $6\frac{1}{2}$ ft. long $\times$ 16 to 18 in. thick. Three No. 12 wrought-iron pipes, each 18 in. diameter, penetrate the water face of the dam, whose upper mouths are protected by a strainer of 2-in. plank anchored to the bed-rock. A separate valve or gate is placed at the lower end of each pipe; the water passing through the three gates, amounting to 280 cub. ft. per second, discharges into a covered timber sluice $7\frac{1}{2}$ ft. wide $\times$ $1\frac{3}{4}$ ft. high, reaching the lower edge of the dam, and opening on to the solid bed-rock of the creek bed. The gates are reached by a man-way above the sluice.

The crest of the dam was formed by a coping of hewn heart cedar timbers, 18 in. wide on top, and fastened securely to the stone wall below by means of iron bolts. In building the lower face wall, care has been taken that it shall be able to resist a large stream of water passing over the crest. In such a case, much water would enter the structure owing to the inclined beds of the face stone and the flat slope of the wall, and would seek escape through the interstices purposely left in the nearly vertical portion of the lower wall. To prevent the consequent hydrostatic pressure, which would accumulate at the base of the dam to perhaps 20 lb. per sq. in., from forcing out the lower face of the wall, the careful building and tying with iron rods was adopted. The dam is built V-shaped, with the vertex of the angle of 15° pointing upstream. The flat slope of 45° was adopted for the walls, as the supply of material was abundant, and it allowed much lighter face-walls to be used.

In connection with the main dam is a waste dam (Fig. 97), which consists of a series of cribs of round cedar timbers, 12 to 30 in. diameter, notched down to heart wood at the joints, and firmly bolted with $\frac{3}{4}$ -in. and 1-in. long drift bolts, the foundation logs being fastened to the bed-rock with $1\frac{1}{2}$ -in. iron dowels. The cribs are solidly filled up with granite rubble of all sizes, from several tons to a few lb., no sand or fine stuff being used. The water-tight lining resembles that on the main dam, being a plank facing of 3-in. heart sugar pine. The crest of the dam is

92½ ft. above datum line ; in it are cut 28 waste-ways, each 4 ft. wide, and 7 ft. deep below the crest. When all danger from freshets has passed, they are closed with boards 3 in. thick × 8 in. wide × 4½ ft. long, placed horizontally, and sliding to their seats one above the other on the incline of the water face. This style of gate is the simplest known, and has been proved by long experience to be the best. The structure is believed to be sufficiently strong to allow a flood of 16,000 cub. ft. of water per second to pass through the wastes and over the crest, without causing damage.

The rather peculiar construction of this dam was due to the following causes:—The stone cliffs in the vicinity of the dam are composed of an exceeding hard granite, with great numbers of short cross seams, thus making it most costly to quarry dimension stone of considerable size. The stone has rarely a good cleavage, and the cost of dressing it down to regular beds is hence great. No limestone is to be found near by, and any lime used must needs have been transported to the work from a long distance. The cost of transport would have been so great as to render the use of lime impracticable.

On the side of the mountain, at the distance of about one mile from the dam, there was a large pile of loose stone, the result of centuries of disintegration of the cliffs above. This stone was too irregular in shape to be used in wall building, but of good quality for an embankment. It was much cheaper to build a tramway to this stone, already quarried by nature, load it on cars, and haul it to the work, than to quarry a smaller quantity from the cliffs nearer the dam.

Hence, the supply of material being abundant, the flat slopes of forty-five degrees for the wall were adopted, which allowed, with safety, very much lighter face walls to be used than would have been the case had they been more vertical.

The stone for the wall as built was quarried from solid rock, and cost, in place, per cubic yard, three or four times more than the loose stone brought from the mountain side. When, in the future, the timber logs forming the cribs in the original 72 feet dam decay, there will be some slight subsidence of superincumbent stone. The depth of the stone is so considerable, and the slopes of the walls so flat, that it is believed this subsidence will not be noticeable.

Ditches.—The two great points to be borne in mind when laying out

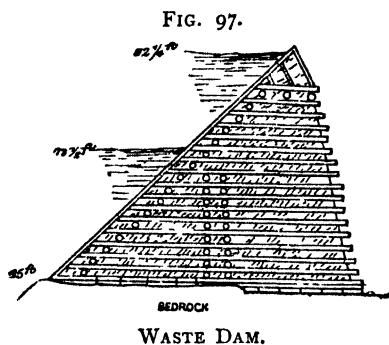


FIG. 97.

a "ditch," are that it shall have a full supply of water all the year round, and that it shall have the greatest possible elevation where it reaches the point at which the water is to be used, the object being not only to supply the diggings in immediate contemplation with water at high pressure, but to be prepared to take advantage of other diggings which may be opened later. The selection of the route to be followed by the ditch is a matter calling forth the greatest skill and judgment of the engineer. By neglect in the first construction, an amount of repairs may be necessitated that in a few years may exceed the original cost of the ditch ; in fact, it is almost impossible to remedy primary defects, and economy in the first instance is always the most expensive in the long run.

An effort should be made to keep the ditch as far as possible between earthen banks, and it must not be forgotten that, until the stream itself

FIG 98.

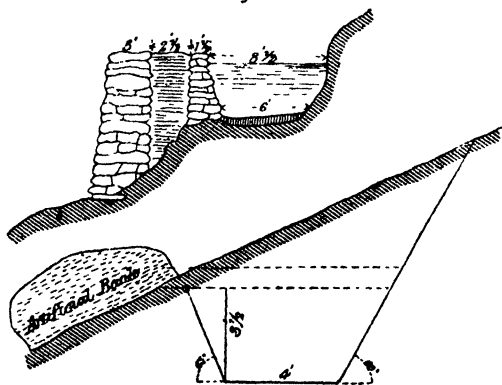
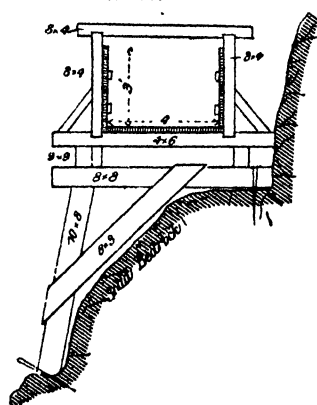


FIG. 99.

DITCHES AND FLUMES.

FIG. 100.



has filled the pores of the ground through which it passes, it is sure to give much trouble. The greatest care must be taken to select ground which is solid, and to avoid such as is likely to produce slides. Often it happens that the nature of the ground will not permit the outside bank to be of earth, in which case masonry may best be substituted, as shown in Fig. 98, or the ditch may be temporarily suspended, and a wooden flume (Fig. 100) run in its stead. In fact, a variety of means may have to be adopted in order to secure the desired line ; it may be necessary to cut a tunnel through a hill, or to employ iron piping in order to cross a valley. The ditch should not be built on very steep inclines, and should if possible, be put so far into the hillside as to have a sufficiently strong bank for its protection outside. The snow-line should at all times be avoided, if possible, as well as those spots where the snow is likely to

drift heavily. Where such are unavoidable, provision must be made for the protection of the ditch, by means of snow-sheds or other suitable covering.

Fig. 99 shows the ordinary mode of constructing a ditch. It is graded in from slope-pegs from 6 to 36 in. All trees within 15 to 25 ft. of the edge of the upper bank are rooted up; also the logs, brush, and leaves from the lower bank, under the artificial bank, are carefully removed.

Fig. 100 shows the method of posting along cliffs, where the foundation is occasionally narrower than the flume. Where flumes connect with the ditch, the posts of the flumes for a distance of several boxes are 4 and $4\frac{1}{2}$ ft. high, allowing an additional side plank. The planking is 2 in. thick.

It is generally preferable to make the ditch deep rather than wide, on account of evaporation. A swift current through a narrow ditch can be made to deliver as much water as a slow current through a wide one, and has the advantage of keeping the ditch free from snow, ice, fallen leaves, and other encumbrances liable to clog it at various seasons. But it is not always possible to observe this rule, for it may happen that, by coming too near the bed-rock, there will be greater leakage than evaporation, so that the reverse plan must be adopted. Great sources of leakage are the holes left by decayed roots, and the filtration of the water beside the roots left in the ground. This is especially the case when the trees found on the line of the ditch are cut down, as is so commonly done, and may be in a great measure remedied by undermining the trees, cutting off the small roots, and rolling the tree, roots and all, down the hillside.

During the summer, the supply of water is often reduced to one-third of its normal amount, a result due to increased leakage and evaporation, as well as diminished rainfall. To overcome the evil as much as possible, all the streams met with on the line of the ditch are made to flow into it as tributaries; but where torrents are encountered, the ditch should be taken over them at a sufficient height to escape the greatest freshets that occur.

As the stream and reservoir supplies combined will probably furnish an excess of water during the wet season, flood-gates must be provided at suitable distances, so as to relieve the pressure, which might endanger the ditch. They also admit of the ditch being rapidly repaired, as the water may be turned out of any section, as required. Overflows must also be constructed, so that a slight flush of water may not injure the banks. All these water outlets must be so arranged and protected as to avoid the possibility of the water undermining the banks of the ditch. The ditch is generally built in sections, which must be connected as rapidly as possible, to be used for the transportation of material. In process of time, the banks of the ditches settle, and become covered in

many instances by a permanent growth of grass and bushes, so that a serious break, even during the severest storms, is very improbable. As to the grade of the ditch, Egleston recommends that 10 ft. per mile should be adopted, as having been proved by Californian experience to be the best on the whole, and one which, while giving the proper velocity, will least endanger the banks. Ditches having a grade of 15 to 20 ft. per mile, delivering 80 cub. ft. per second, have been successfully used; but they are exceptional. The grade, once determined, must or should be adhered to throughout.

"In locating the route of a projected ditch and selecting the source of its water supply, both should be maintained at as high a level below the deep snow line as possible; another point to be held in view being the securing of an ample summer stock of water, including facilities for its storage. If kept at a high elevation, the ditch will command a larger scope of mining territory, besides being able to deliver the water under increased pressure, a condition on which its effective power is largely dependent. The ditch, if too low, loses these advantages; if too high, it is liable to suffer obstruction from snow and ice, delaying the season of active operations, and necessitating considerable expense for freeing it from these impediments. With a given amount of water, more and better work can be done in warm than in cold weather. Among secondary objects to be kept in view, during the preliminary stages of these enterprises, is the securing, by location or purchase, of all streams along the main trunk that can readily be made tributary thereto. By attending to this, any waste occurring through evaporation, absorption, or leakage, may be made good, and the water flow thus be kept to the full carrying capacity of the ditch; which latter, where such addition to the water is feasible, may, sometimes to great advantage, be enlarged along its lower portions, where the difficulties of construction are apt to be less than they are higher up. These subsidiary streams may also be serviceable in helping to replenish the main ditch when its principal sources of supply begin to fail, as they usually do with the advance of the dry season. Where it is not desirable to empty the water of these side streams into the main ditch, it is to be carried over or under the latter, as found most convenient. Because of the rapidity with which evaporation goes on in the warm and arid climate of California, deep and narrow ditches, other conditions being equal, are preferable to broad and shallow ones. Where, however, the former involve the necessity of much extra bedrock cutting, economic considerations require that they should give place to shallow ditches made wider." (De Groot.)

The following table will illustrate the degree to which ditches differ in their proportions.

Name.	Length of Ditch.	Width of Top of Ditch.	Width of Bottom of Ditch.	Depth of Ditch.	Cost of Ditch.	Average Grade per Mile.	Discharge in Miners' Inches.
	miles.	ft.	ft.	ft.	£		
Milton	100	6	4	3.5	52,000	14.5	3000
North Bloomfield	55	8.65	5	3.5	84,000	14	3200
.. .. . (1.)	60	8	6	4	..	..	2200
Spring Valley (2.)	52	6	4	3.5	..	..	2000
Hendrick's	46.5	6	4	2	27,000	9.6	..
San Juan	45	..	..	..	59,000	..	1300
South Yuba (3.)	35	8	4	4	..	..	..
Excelsior	33	8	5	4	..	9	1700
La Grange (4.)	20	9	6	4	90,000	7.5	3000
Eureka Lake	18	..	..	..	86,000	..	2800
Union	15	8	4	3.5	..	13	1200
Boyer	15	8	4	3.5	..	13	1200
.. .. . (5.)	..	6.5	4	3	..	11.2	3000
.. .. . (6.)	..	..	3	3	..	..	2000

1. On the line of the ditch are 4 miles of iron pipe, 30-in. diameter, one section of which conducts the water across a branch of a river. It is laid as an inverted syphon, and has a vertical depression of 856 ft. The receiving arm has a head of 180 ft. vertical pressure; length of syphon, 24 miles.

2. This has 34 miles of 30-in. iron pipe.

3. With a subsidiary ditch, grade 10 ft. per mile, current 2½ ft. deep. It is carried across a narrow cañon by a wire suspension flume, and across another by a truss flume with a span of 60 ft.

4. Most of this ditch is hewn in granite.

5. The line is graded 7½ ft. wide, and excavation made close into the bank, leaving not less than 1 ft. of solid earth on the outside.

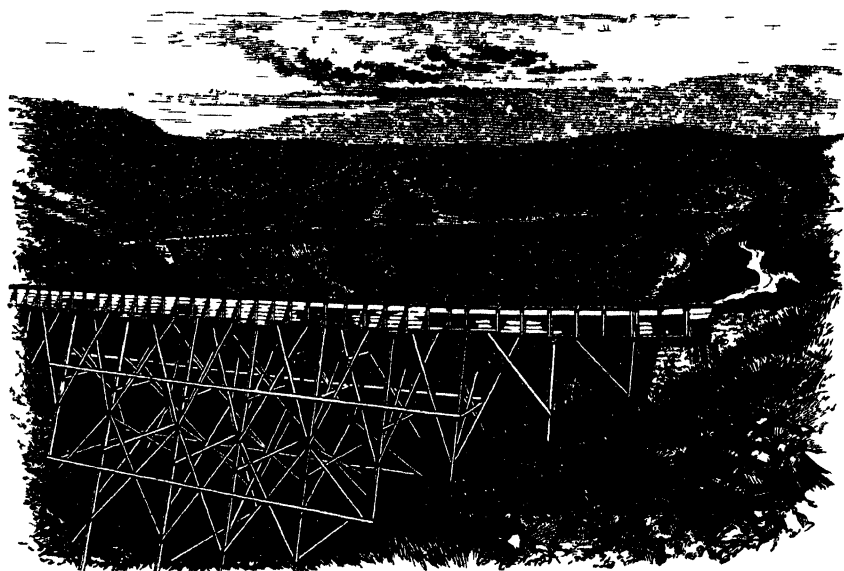
6. To cross a creek the water is led into a 27-in. pipe, 420 ft. long, and with a depression and elevation of 75 ft.

Flumes.—It sometimes happens that the ditch must cross ravines of some depth, or be carried over a long stretch of level ground, in order to deliver the water at the proper height. In such cases, wooden flumes or launders are often used to conduct the water. Their employment should be avoided on all possible occasions, as they are very expensive to build, need constant repair, are very liable to burn down in summer, and in winter are very likely to freeze solid, especially where they are exposed to all winds. Formerly they were much more favoured than now, because wood was plentiful and cheap, iron dear and difficult of transport; but no engineer nowadays would think of copying such gigantic and expensive structures as that built near Smartsville, and which has served to adorn numberless pages. Nevertheless, smaller constructions of the kind are a common feature in the mining districts, despite their objections. They are generally made smaller than the ditch, and therefore of much higher grade, in order to accommodate the same quantity of water. A common grade is 30 to 35 ft. per mile, and they should be set in as straight lines as possible; if a curve must be made, it should be of very wide radius. The utmost attention should be paid to the foundations, and to placing the uprights so that they shall not settle, as nothing is so detrimental to a proper water-supply as a change of grade in the flumes through settling. Where the grade changes, the water is almost sure to freeze solid, and thousands of feet have sometimes to be cut out for that reason. The posts should be placed above the ground, so as not to rot; the planking is 12 in. to 18 in. wide and 1½ to 2 in. thick. The joints are covered on the outside by

strips 3 in. wide and $1\frac{1}{2}$ in. thick. The flume is supported at every 4 ft. by a framing of 4 in. $\times$ 4 in. timber ; the sills in the heavy flumes extend 18 in., and are braced to the side, the lighter ones are braced to the uprights. An example erected in New Zealand is shown in Fig. 101.

When very high, flumes should be anchored with wire rope, as a safeguard against swaying with the wind. They should be built in an open country, if possible, as a protection against forest fires ; and brushwood and other inflammable material should be removed from their vicinity. When it is impossible to put them in open country, they should be carried as close as possible to the bank, so that, in the event of snow-slides, &c., the snow may pass over, and not against them. This latter

G 101



SCANDINAVIAN WATER RACE, NEW ZEALAND.

contingency requires in some places that very strong "snow-guards," or overhanging sheds, be built over the flume. Under the best conditions, free from ordinary accidents, flumes will not last longer than 10 to 15 years. This is emphatically the case where they run dry during a portion of the year, for it is a well-known fact that wood which is sometimes wet and sometimes dry will last a much shorter time than that which is always wet or always dry. The alternate expansions and contractions destroy the fibre of the wood, and start the nails and wedges. The average cost of repairs to a flume will almost double that of an equal length of ditch.

The following tables, published by Joshua Hendy, San Francisco, will be useful in calculations pertaining to canals, ditches, and flumes.

TABLE NO. I.

FLOW OF WATER IN OPEN CHANNELS—BASE TO PERPENDICULAR OF THE SIDE SLOPES BEING AS 3:4.

Fall per Mile.	Fall per Rod.	T 2.2 ft. B 1.0 ft. D .8 ft.	T 3.3 ft. B 1.5 ft. D 1.2 ft.	T 4.4 ft. B 2.0 ft. D 1.6 ft.	T 5.5 ft. B 2.5 ft. D 2.0 ft.	T 6.6 ft. B 3.0 ft. D 2.4 ft.	T 7.7 ft. B 3.5 ft. D 2.8 ft.	T 8.8 ft. B 4.0 ft. D 3.2 ft.	T 9.9 ft. B 4.5 ft. D 3.6 ft.	T 11 ft. B 5 ft. D 4 ft.	T 13.2 ft. B 6.0 ft. D 4.8 ft.	T 16.4 ft. B 7.0 ft. D 5.6 ft.	T 17.6 ft. B 8.0 ft. D 6.4 ft.	T 19.8 ft. B 9.0 ft. D 7.2 ft.	T 22 ft. B 10 ft. D 8 ft.
		Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.	Section sq. ft.
Feet.	Inches.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.
1	.0375	.45	1.33	2.67	5.57	9.05	13.46	20.26	28.04	37.1	58.4	96.5	138.3	189.2	261.2
2	.0750	.63	1.88	3.87	7.88	12.80	19.04	28.64	39.67	52.4	82.7	136.4	195.7	267.6	369.4
3	.1125	.77	2.30	4.74	9.65	15.67	23.32	35.08	48.59	64.2	101.4	167.1	239.6	327.7	451.3
4	.1500	.89	2.65	5.47	11.14	18.52	26.93	40.51	59.10	74.1	117.1	192.9	276.7	378.4	522.3
5	.1875	1.00	2.97	6.12	12.46	20.24	30.11	45.30	62.71	82.9	130.9	215.7	309.3	423.1	584.0
6	.2250	1.09	3.25	6.70	13.65	22.17	32.98	49.62	68.70	90.8	143.4	236.3	338.8	463.5	639.8
7	.2625	1.18	3.42	7.24	14.74	23.94	35.63	53.58	74.19	98.1	154.8	255.3	366.0	500.5	691.0
8	.3000	1.26	3.75	7.73	15.75	25.60	38.08	57.28	79.53	104.8	165.5	272.9	391.2	535.1	738.7
9	.3375	1.34	3.98	8.21	16.71	27.15	40.39	60.76	84.14	111.1	175.6	289.4	415.0	567.6	783.5
10	.3750	1.41	4.19	8.65	17.61	28.62	42.57	64.05	88.68	117.1	185.1	305.0	437.4	598.2	825.9
11	.4125	1.48	4.40	9.07	18.47	30.02	44.65	67.18	93.02	122.9	194.1	319.9	458.7	613.2	866.2
12	.4500	1.54	4.60	9.48	19.30	31.35	46.64	70.65	97.15	128.4	202.8	334.2	479.1	655.4	925.6
13	.4875	1.61	4.78	9.86	20.08	32.63	48.54	73.03	101.13	133.6	211.1	347.8	498.7	682.1	941.7
14	.5250	1.67	4.96	10.24	20.84	33.87	50.38	75.79	104.94	138.7	219.0	360.9	517.5	707.8	977.2
15	.5625	1.73	5.14	10.60	21.57	35.05	52.14	78.44	108.63	143.5	226.6	373.6	535.7	732.8	1011.5
16	.6000	1.78	5.31	10.94	22.27	36.20	53.86	81.02	112.18	148.2	234.1	385.9	553.3	756.7	1044.7
17	.6375	1.84	5.47	11.28	22.96	37.31	55.51	83.51	115.64	152.4	241.3	397.8	570.3	780.1	1076.9
18	.6750	1.89	5.63	11.60	23.63	38.39	57.11	85.93	118.99	157.2	248.3	409.3	586.9	802.7	1108.1
19	.7125	1.94	5.78	11.92	24.28	39.44	58.58	88.29	122.26	161.5	255.1	420.5	601.5	824.8	1138.4
20	.7500	1.99	5.93	12.23	24.91	40.47	60.21	90.58	125.43	165.7	261.7	431.4	618.5	846.1	1168.0
21	.7875	2.04	6.08	12.54	25.53	41.47	61.70	92.82	128.53	169.8	268.2	442.0	633.9	867.0	1196.8
22	.8250	2.09	6.22	12.83	26.12	42.45	63.15	95.00	131.55	173.8	274.5	452.5	648.8	887.4	1225.0
23	.8625	2.14	6.36	13.12	26.71	43.40	64.57	97.15	134.51	177.7	280.7	462.9	663.4	907.4	1252.6
24	.9000	2.18	6.50	13.40	27.29	44.34	65.95	99.23	137.40	181.5	286.7	472.6	677.7	926.9	1279.5
25	.9375	2.23	6.63	13.68	27.98	45.24	67.32	101.28	140.24	185.3	292.6	482.3	691.6	946.0	1306.0

T signifies top width; B, bottom width; D, depth.

TABLE No. 2.

FLOW OF WATER IN OPEN CHANNELS—BASE TO PERPENDICULAR OF THE SIDE
SLOPES BEING AS 2 : 1.

Fall per Mile.	Fall per Rod.	T 6 ft. B 2 ft. D 1 ft.	T 9 ft. B 3 ft. D 1.5 ft.	T 12 ft. B 4 ft. D 2 ft.	T 16 ft. B 6 ft. D 2.5 ft.	T 22 ft. B 10 ft. D 3 ft.	T 28 ft. B 12 ft. D 4 ft.	T 40 ft. B 20 ft. D 5 ft.
		Section 4 sq. ft.	Section 9 sq. ft.	Section 16 sq. ft.	Section 27.5 sq. ft.	Section 48 sq. ft.	Section 80 sq. ft.	Section 150 sq. ft.
Feet.	Inches.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.	Cu. ft.
.5 ..	.01875	1.27	3.85	8.63	18.11	38.79	78.2	188.1
.6667	.0250	1.46	4.44	9.96	20.91	44.79	90.3	217.2
.8333	.03125	1.63	4.96	11.14	23.38	50.08	101.0	242.8
1 ..	.0375	1.79	5.44	12.20	25.61	54.86	110.6	266.0
1.25 ..	.046875	2.00	6.08	13.64	28.68	61.32	123.7	297.4
1.5 ..	.05625	2.19	6.67	14.96	31.34	67.26	135.7	326.1
1.75 ..	.065625	2.37	7.19	16.14	33.88	72.57	146.4	351.8
2 ..	.0750	2.53	7.69	17.26	36.22	77.58	156.5	376.1
2.25 ..	.084375	2.68	8.16	18.30	38.42	82.29	165.9	399.0
2.5 ..	.09375	2.83	8.60	19.29	40.50	86.72	174.9	420.6
3 ..	.1125	3.10	9.42	21.14	44.36	95.00	191.6	460.7
3.5 ..	.13125	3.35	10.17	22.83	47.91	102.60	207.0	497.6
4 ..	.1500	3.58	10.87	24.41	51.22	109.70	221.3	531.9
4.5 ..	.16875	3.79	11.54	25.88	54.33	116.30	234.7	564.2
5 ..	.1875	4.00	12.16	27.29	57.27	122.70	247.4	594.8
6 ..	.2250	4.38	13.31	29.89	62.74	134.40	271.0	651.5
7 ..	.2625	4.73	14.39	32.29	67.78	145.10	292.7	703.6
8 ..	.3000	5.06	15.38	34.52	72.43	155.20	312.9	752.2
9 ..	.3375	5.37	16.31	36.61	76.83	164.60	331.9	797.9
10 ..	.3750	5.66	17.19	38.59	80.99	173.50	349.9	841.1
11 ..	.4125	5.93	18.03	40.47	84.94	181.90	366.9	882.1
12 ..	.4500	6.20	18.74	42.27	88.72	190.10	383.2	921.3

T signifies top width; B, bottom width; D, depth.

TO DETERMINE THE CAPACITY OF DITCHES, CANALS, OR FLUMES.

In cases where the sectional areas are equal, but of different forms:

The simplest form of construction of a ditch or canal is that in which the width of the bottom is made equal to one of the sides, and in which the base to the perpendicular of the side slope is as 3 : 4; and this form has been adopted as the standard where the topography of the ground will admit.

The relative carrying capacity for trapezoidal form, base: depth of slope :: 3:4; bottom width: depth :: 5:4. Coefficient of capacity = 1000.

Trapezoidal form, base: depth of slope :: 1:1; bottom width = depth, .994.

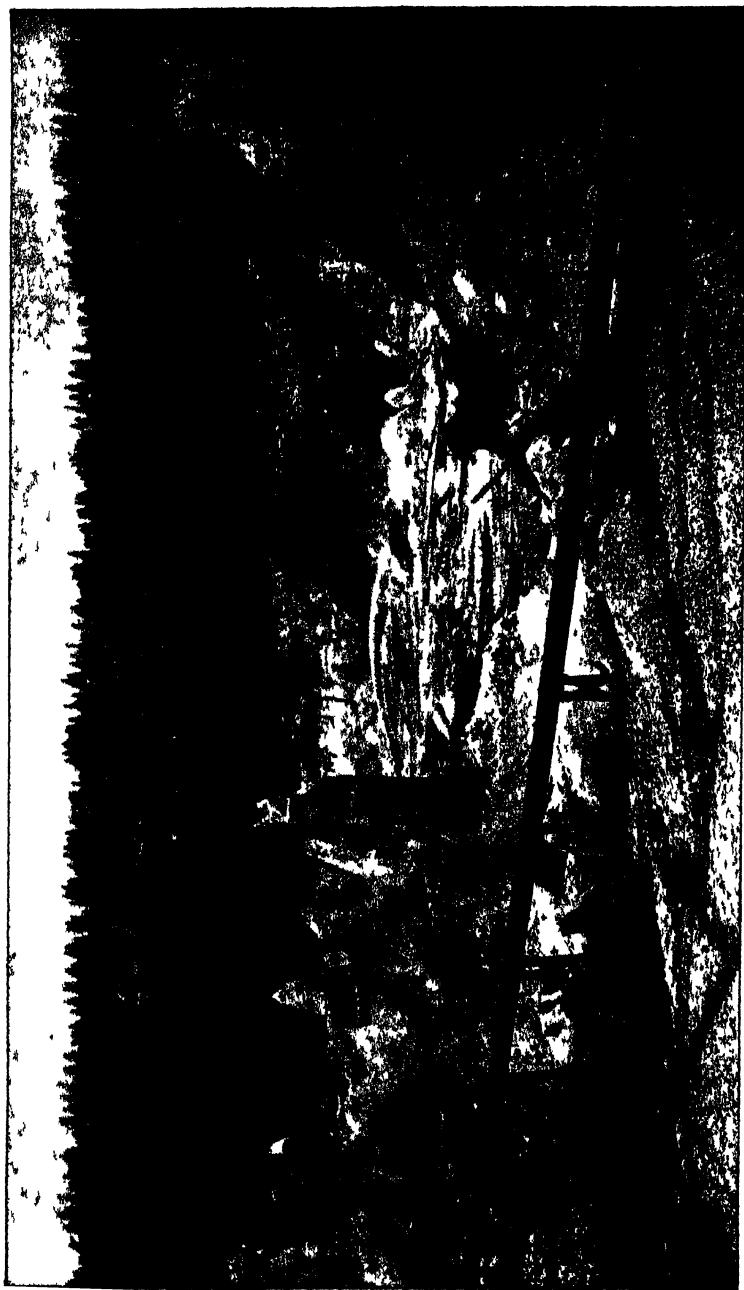
Flume, 2:1, .961; semi-hexagonal, 1.008; square, .925; semi-circular, 1.056.

The fall being 8 feet to the mile, and the sectional area of a square flume being 11.52 square feet, what will be its carrying capacity per second?

In Table No. 1, in column headed "fall per mile," find the given fall 8 feet, and opposite in column headed (6' 6"-3'-2'-4") or a sectional area of 11.52 square feet will be found 25.60 cubic feet, which, multiplied by the coefficient for a square given above, gives $25.60 \times .925 = 23.68$ cubic feet, or $\times .50 = 11.84$ miners' inches.

The tables given to determine the flow of water through ditches, canals, or flumes have been computed upon the assumption that they are generally smooth and straight.

Pipes.—There is no doubt that wherever it is possible to do so, wooden flumes may best be supplanted by iron pipes. These are generally made of No. 12 to No. 16 sheet-iron, in variable lengths, which are joined together when placed in position on the ground. At first the "length" was restricted to 12 ft., but now 15 ft., 20 ft. and even more are common figures. These pipes are very light in comparison with their strength, and have the important advantage of being easily removed,



E. & F. N. Spon. London & New York

IRON PIPE

NORTH BLOOMFIELD HYDRAULIC MINE, NEVADA COUNTY, CAL

NK PHOTO BRADLEY & CO LONDON

by simply cutting the joints and rejoining them in their new position. Repairs are facilitated by the same reason, and a few extra lengths are always kept on hand so as to avoid delays. The pipes are best buried in the earth, at a sufficient depth to escape the expanding and contracting effects of the weather; when above ground, they are borne on trestles. Sometimes it is inconvenient to carry a flume across a place, or to run the pipe as a siphon, in which case the pipe may be led across on frames, built like a flume, with only a floor and a top board. The pipes are generally riveted together, and will stand very great pressure, more frequently collapsing than bursting.

The Spring Valley Water Co. has 17 miles of 30-in. riveted wrought-iron pipe, which carries water across depressions of 150 ft., 200 ft., and 250 ft. in vertical height; and 14,000 ft. of 30-in. pipe was laid across a valley nearly 1000 ft. deep. In such a case, the water must be discharged into a head-box at a point high enough above the discharge-level to give pressure sufficient to overcome the friction, which in this instance was estimated at 20 ft.

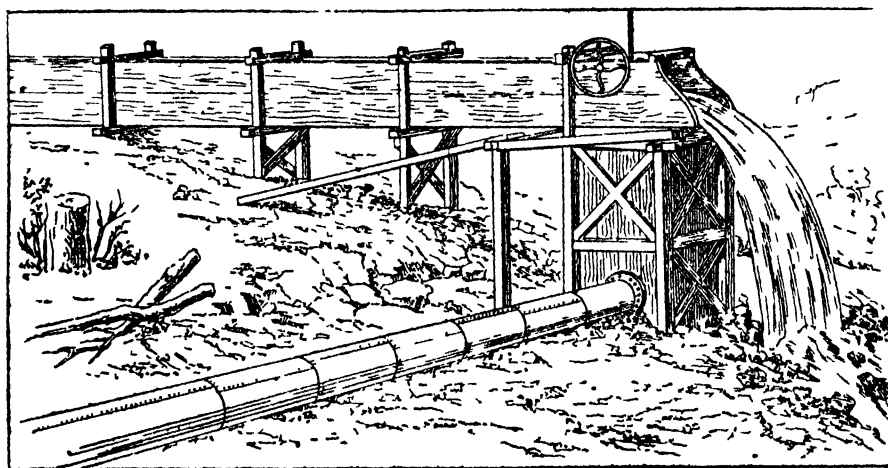
Raymond mentions 2 miles of pipe made of sheet iron, in 20-ft. sections, slipping one into another, and tapering from the reservoir to the workings from 22 in. to 8 in. In another case, the joints were fastened together by a riveted sheet-iron collar, made of No. 16 iron 3 in. wide, daubed over with melted asphaltum, slipped over the joints and held in place by the adhesion of the asphaltum. As originally constructed, a portion of the pipe was intended to withstand a pressure of 650 ft. in crossing a sag in the mountain; but when the water was turned on, the pressure forced it all out of the joints. Leaking the joints was next tried, but with no better success; and as the efficacy of an inside collar, as usually applied in such cases, was not known, the pipe had to be taken up and run round the mountain.

The same authority gives some details of the construction of another pipe, which cannot fail to be interesting. The inlet to the pipe was 150 ft. above the outlet, with a vertical height of 900 ft. from the lowest point to grade line. The pipe was 30 in. diameter, and intended to carry 1900 miners' "inches" of water. The thickness of iron used was No. 14 for 150-ft. pressure, No. 12 for 275-ft., No. 10 for 350-ft., No 7 for 425-ft., $\frac{1}{4}$ in. for 600-ft., $\frac{1}{8}$ in. for 850-ft., and $\frac{3}{8}$ in. for 900-ft. The water was admitted at the upper end from a cistern with sand-box, &c., for settling any sand or gravel brought in from the ditch. Here the pipe had an elbow dipping into the water, to prevent the entrance of air, and at 50 ft. from the inlet there was a standpipe to permit the escape of any air accidentally imprisoned. The pipe was laid in a trench 5 ft. deep, and covered with earth to prevent undue expansion and contraction in hot and cold weather. The sections were 23 ft. long, and

were riveted one to another continuously, manholes being placed at every 1000 ft. to allow of the entrance of the workmen. The rivets used were for No. 14 iron, $\frac{1}{4}$ -in. wire ; No. 12, $\frac{1}{4}$ -in. ; No. 11, $\frac{5}{16}$ -in. ; No. 9, $\frac{3}{8}$ -in. ; and No. 7, $\frac{3}{8}$ -in., driven cold. The $\frac{1}{4}$ -in. iron was machine-riveted cold, hand-riveted hot ; $\frac{5}{16}$, $\frac{5}{8}$, $\frac{3}{4}$, $\frac{3}{4}$ driven hot. A steam riveting-machine was principally used, and found much superior to hand. The $\frac{3}{8}$ -in. *wrought* iron here sustains a pressure of 385 lb. per sq. in., which would require nearly 3-in. *cast* iron.

The supply-pipes discharge into a "head-box," "pressure-box," or "bulkhead," such as is shown in Fig. 102, and which is often also the measuring box. This box, which is made of great strength, is usually about 8 ft. deep, and is provided with an iron grating, to keep out floating

FIG 102



HEAD-BOX, PRESSURE BOX, OR BULKHEAD.

matters. The greater the depth of water in the bulkhead over the mouth of the pipe, the less air can enter it. There should never be less than 4 ft., and 5 or 6 ft. is much better.

"A sand box is frequently put into the bed of the ditch which conveys the water to the mines, at several points along its course, according to the quantity of impurity present. When so used, or when combined with the pressure box, its construction is the same. It is so arranged that it will arrest and retain sand and small gravel that may be held in suspension and carried along by the force of the current. The sand box varies in dimensions with the capacity of the ditch. It not only extends across the whole bed of the ditch, but is wide in proportion to the amount of sand which the water contains. When required to be of extraordinary size, it is placed outside the ditch, with which it is connected by flumes,

which take the water from, and return it to, the ditch, after it has been divested of its sandy impurity. When placed in the ditch, it is several feet below the bed, forming a depression into which the sand falls. The sand box is provided with a door or gate, through which the accumulation may from time to time be sluiced out.

"After the water from a mining ditch has been used and returned to it at a lower level, which is frequently the case, it is so highly charged with sand and finely divided suspended mineral matter, that it is found necessary to purify it by mechanical means. This is done by running it into large tanks or reservoirs, generally natural depressions cut by mountain streams, across which a dam has been built. The water having been let into the reservoir, is allowed to settle, and the water is drawn from the lower end in the same sized stream which enters at the upper.

"The water in the pressure box must be without motion before it enters the pipes, or air will be carried down, which will cause a disturbance and produce an intermittent stream at the nozzle. To secure this quiet, the water is received into one apartment—which serves also for a sand box—from which it flows through a lateral opening into the pressure-box.

"In some cases, the pressure box for a single pipe is divided into three compartments, built of 2-inch plank in a frame-work of 10 inch timbers. The water falls into the outer chamber or sand box, the bottom of which is lower, to admit of the settling of the sand and small gravel. When the water rises to the level of the second compartment, it passes through a vertical grating of round iron bars $\frac{1}{2}$ inch in diameter, set 3 inches apart. This grating intercepts all pieces of wood and other floating rubbish. From the centre compartment, the water flows into the pressure box through a similar grating, in which the rods are set $\frac{1}{2}$ inch apart, and catch the leaves and smaller floating bodies that may have passed the coarser grating.

"From this third compartment the iron pipe emerges, being flanged and spiked to the plank inside. There being no pressure here, the iron is light. When working properly, all the boxes are full of water, which enters the iron pipe quietly and without air. The slight excess flows over a depression in the upper edge of one of the compartments, which is a shallow water-gate; but when the water is paid for, this waste is made as small as possible, by closing the gates which supply the measuring box.

"When two pipes lead from the same pressure box, the sand box is sometimes placed in the centre. The usual size of the sand box is 10 by 20 feet in the single, and 12 to 30 feet in the double. There are much larger boxes in use, which combine the pressure, sand, and measuring box in one." (De Groot.)

The feed-piping is made sometimes of canvas, sometimes of iron : canvas hose is only used in accessible places, or on very small claims, and is made of heavy duck sewn together. With ordinary sewing, it will bear a pressure of 50 ft. perpendicular without other support. When greater pressure is needed, it is surrounded by iron rings, about 3 in. apart, and connected by cords, then called "crinoline" hose, and equal to a pressure of 180 ft. of water. Failing iron rings, a netting of $\frac{1}{4}$ -in. or $\frac{1}{2}$ -in. rope, with meshes about 2 in. sq., may be used, and is almost equally strong. Pipe of this description can be easily shortened, lengthened, repaired, and transported, and seemed, at the time of its invention, to be the perfection of a water-conduit ; but it was not long before iron piping generally supplanted it.

The ordinary dimensions of iron feed-pipes are as follow :—

Diameter of Pipe.	Pressure.	Number of Iron.	Thickness of Iron in decimals of an inch.
in.	ft.		
22	150	16	0·060
22	150 to 250	14	0·078
22	250 to 310	12	0·098
30	150	14	0·078
30	150 to 275	12	0·098
40	160	..	0·236

The iron used varies generally from No. 16 to No. 11, according to the pressure, the best iron only being employed. The size of the pipe will depend upon the supply of water ; with 1500 to 2000 in. of water, a 22-in. pipe will suffice ; where the supply is 3000 in., a 30-in. pipe must be used, and so on.

The feed-pipe should pass from the bulkhead to the workings in as direct a line as possible, and upon an even grade, and should be furnished with air-valves about 6 in. in diameter at about every 100 ft. of its length. These valves are of paramount importance, for in case there should be any depression in the pipe, the water would be retained there, and if it should be discharged faster than it can enter the pipe, a collapse must take place, unless the valves are in their places and working to prevent the formation of a vacuum. They should be of brass, and never of wood, for the latter have been known to swell, and become so tight as not to act, resulting in a collapse of the pipe, against which every precaution should be taken, as the least inconvenience it can occasion will be a serious delay.

The feed-pipe terminates in a distributor, generally a cast-iron box provided with openings, to which are attached the pipes carrying the nozzles. These openings are closed by ordinary water-gates, which are

worked through a screw on the outside, by which the water is supplied to or shut off from any one of the nozzles. The distributors have as many as 4 gates when the water-supply is adequate for 4 nozzles from a single box. When the site for the distributor has been chosen, it is permanently fixed there. The pipes leading from the distributor to the nozzles are 10 in. to 15 in. in diameter, made of No. 14 to No. 12 sheet iron in 12-ft. lengths, composed of sections, 18 in. long, riveted together. The lengths, or "boxes," as they are also called, in imitation of the 12-ft. lengths of the sluices, are occasionally connected by flanges, or by hooks and wire, and in some cases are simply telescoped together like stove-piping. Another plan is to use bolts and nuts, so that the structure may be easily set up or taken apart. In all cases, the joints must be made quite tight. The pipe is generally supported so as to avoid bends; if laid on the ground, great attention must be paid to the air-valves, and each "box" should be well painted with coal-tar. The miners always repair their own pipes, a workshop and appliances being maintained for the purpose.

A recent report, by the State Mineralogist of California, contains the following remarks on pipes.

No. 14 iron will resist a pressure of 300 feet head, or 130 lb. to the square inch, and an 11-inch pipe, of No. 16 iron, a pressure of 500 feet, or 217 lb. to the inch. No. 14 iron is 0.083 inch thick, and weighs 3.35 lb. to the square foot; No. 16 is 0.065 inch thick and 2.63 lb. to the foot. Persons having no practical experience generally make their pipes unnecessarily heavy.

The manufacture of iron pipe is very simple. American iron is used. The sheets are taken to a machine which punches all the holes at a single motion, and are then curved, lapped, and riveted. The joints are from 24 to 30 inches long; the rivets used are proportional to the size of the pipes. The short lengths are then riveted together in lengths of 15 to 28 feet. The short joints are slightly tapered, so that the small end of one will fit into the large end of another. The long sections are dipped into a bath of hot asphaltum; the pipes are lifted by machinery and allowed to remain in the bath until they have acquired the temperature of the melted asphaltum. It is claimed that a pipe so prepared will last under ground, for 20 years, and this may be regarded as the extreme term of usefulness for first class hydraulic pipe. A single row of rivets is generally all that is required, but in some cases, where great strength is desired, a double row is used.

It has been found by experience that it is not necessary to rivet the sections of pipe together; in fact, unless it can be protected from changes in temperature, a continuous pipe many hundreds of feet in length, and strongly riveted together, would be subject to an inconvenient movement

if not in continual danger of breaking ; the joints are simply slipped into each other like a stovepipe, until the line is as long as required. The wide ends are toward the pressure boxes ; the ends are entered, and a rope strap is put on each section ; a strong tackle is hooked on, and the lengths are drawn together ; one strand of marline is then caulked in, and the edges are hammered down.

Iron pipes expand and contract with every change of temperature. In a hot day it can be seen where the joints have slipped several inches. In some parts of the pipe this contraction is so great as to endanger the pipe. In such a case an iron band is used, to give strength and at the same time to admit of expansion and contraction, so arranged that the slip does not cause a leak. These bands are from 3 to 12 inches wide. Several plans have been adopted to draw them tight. There is always a bandage of canvas or gunny sack laid under the iron band ; the common method of fastening is by set-screws or wedges. The system by screws is the most convenient to put on, but the threads soon become rusty, and the screws are difficult to remove, while if a wedge is used it can be struck out by a blow from a hammer, no matter how rusty it may have become.

The expansion becomes at times quite annoying, and, when convenient, the pipes are covered either with earth or boards, to protect them against the sun's heat. Rough boards are also required to protect the pipes against falling rocks thrown up in blasting boulders. According to its degree of inclination, the pipe requires to be firmly braced, and at certain points to be weighted down with rocks, a secure foundation being a desideratum.

When water is let into the pipes, air is displaced, but sometimes collects in the highest elevations. This causes the pipe to undulate, or to "buck," as it is technically called. To allow the air to escape, an ingenious valve has been devised. At regular intervals on the upper curved surface of the pipe, oval holes are cut 3 by 4 inches. On the inside of the pipe, valves of sole leather, strengthened by plates of iron or brass, are so attached that a pressure from below forces them up, closing the hole. The hinge is at the upper end of the hole, or that part nearest to the source of the stream. The flow of water shuts the valve, and the pressure holds it in place. To allow the air to escape, a weighted lever extends lengthwise along the pipe, which carries a wooden pin or dowell set against the closed valve. The weight is so regulated as to force down the valve against the pressure of air, but to yield to that of the water. As long as the air only acts on the valve, the dowell forced down by the weighted box opens the valve and allows the air to escape. When the pipe is full of water, the valve and weight become inoperative. Every change in the position of the nozzle causes the pipe to undulate.

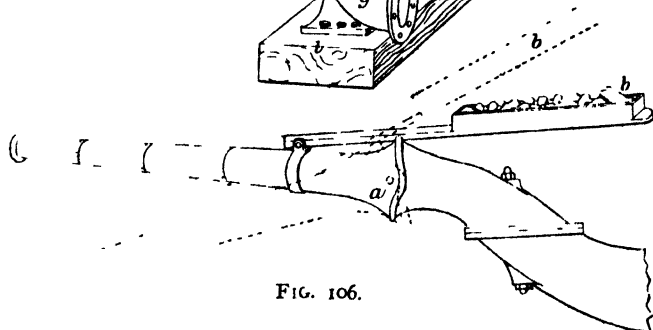
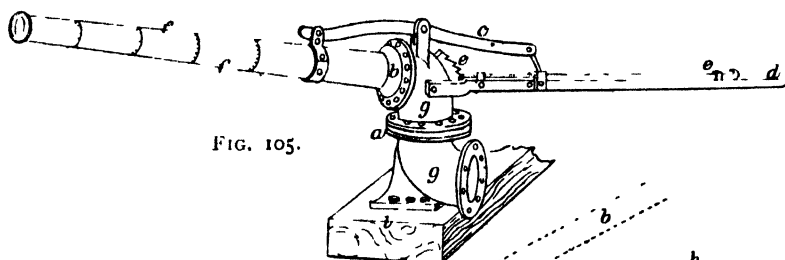
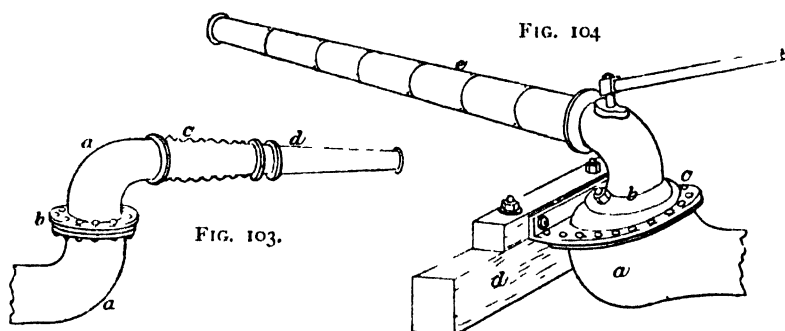
The effect is shown at the pressure box, and the gates have to be adjusted. The pipes should be kept well painted. The pipes of the Polar Star and Southern Cross, at Dutch Flat, are in a remarkably good state of preservation, which is mainly owing to the fact that they are protected by heavy coats of paint.

The distributor, formerly in common use, consists of a strong iron box, which, on receiving the water from the supply pipe, passes it out through as many apertures in its several faces as there are streams to be played in the mine, four of which sometimes issue from a single distributor. To each of these apertures there is attached a smaller iron pipe for conducting the water to the several nozzles. The water is shut off from these pipes by iron gates. The distributor is only useful with a small head. The enormous pipes now in use, carrying several thousand inches of water under a pressure of hundreds of feet head, cannot be shut off from below. Nothing in use would stand such a shock or pressure. When it is required to shut off the water, it must be done at the head. For this reason, the distributor has gone out of use, and has been replaced by the "twin" and the "triplet." The "twin" is a fork in the pipe, by which a portion of the water is conducted to one point and a portion to another. The two branches are used at alternate intervals each supplying a nozzle in different parts of the claim, or two nozzles are supplied at the same time. In the latter case, the branch pipes are of half the capacity of the supply pipe; and in the former, both the supply pipe and the two branches are of the same diameter. In both cases there are strong gates in both branches, closed from above by a screw. The "triplet" has three branches. What has been said of the "twin" applies equally to the "triplet." The "twins" and "triplets" are weak points in the pipe, and require to be especially strengthened. The joints are not only braced by strong timbers, but are weighted down by heavy pipes of stones.

Nozzles.—Passing mention has already been made of "nozzles." Their use is to concentrate the water so that it shall be discharged against the bank with sufficient force. Several kinds of nozzle have been introduced from time to time, one of the first being that shown in Fig. 103, and known as a "goose-neck." It consists of two elbows of iron pipe *a*, working one over another, with a flexible joint *b* between them, which admits of a certain amount of rotation in a horizontal plane; while the piece of hose *c*, inserted between the upper elbow and the nozzle *d*, permits a moderate vertical movement. The facility of moving in all directions constitutes a great improvement on the straight nozzle; but the sharp double bend of the elbow reduces the force of the water, and the flexible joint is liable to get hard and "buck."

Craig's "monitor," Fig. 104, was devised to remedy these defects.

Its principal parts are : a hollow globe *a*, provided with two openings, one at the side for admitting the main supply-pipe, and a second on the top for feeding the nozzle. The latter has an elbow with a hemispherical attachment *b*, turned so as to exactly fit the inside of the flange *c*, which is bolted to the globe, and, with the leather packing, makes a water-tight joint. The nozzle is thus movable in any direction at will, to the extent of nearly 40° vertically and 360° horizontally. The apparatus is bolted firmly to the timber structure *d*. The nozzle is composed of a wrought-iron tube *e*, 10 ft. long, to the end of which is screwed the nozzle proper, made of cast iron, and 6 to 8 in. inside diameter. When the direction has been determined, the nozzle is commonly tied in place, but with 180-ft. pressure, it needs no support.



HYDRAULIC HOSE NOZZLES.

Fisher's "knuckle-joint," Fig. 105, consists of two elbows *g*, placed together in the form of an *S*, and arranged to rotate horizontally by means of a ring-joint *a* furnished with friction-rollers. The ring is

slipped down over the lower elbow, and held in place by a flange on the pipe; then bolted to a flange on the top elbow, joining the two, but allowing the latter free action. Indiarubber is used to make the joints water-tight. The nozzle is secured to the upper elbow by means of a knuckle-joint *b*. Movement in all directions is easy, and can be instantaneously effected, vertically by a series of levers *c d*, the main arm of which is 10 to 12 ft. long, and held in place by a ratchet *e*. The elbow and knuckle-joint are of cast iron, $\frac{5}{8}$ in. to $\frac{3}{4}$ in. thick; the pipe *f* is of No. 16 sheet iron, 8 ft. long, the nozzle of cast iron. The pipe does good work to a range of 200 ft., and keeps its position when in use.

Two forms of nozzle have been invented by Hoskins, and both are extensively employed. The earlier form was the "dictator," which differed from Craig's monitor principally in that the joint was external instead of internal, causing the water to have a tendency to force the joints apart rather than together. Indiarubber packing is used in the joint, and friction wheels are introduced to reduce the friction, so that the pressure being all against elastic packing, the nozzle is easily managed, and its movements are easy. It has the same motions as Craig's, but its construction is more complicated, and the sharp bends of the joint cause a loss of power. The later and most popular form is the "little giant," shown in Fig. 106. It is very simple, easily repaired, has no sharp curves, and is said to discharge the greatest amount of water with the least resistance. Horizontally it rotates completely, and moves vertically on the knuckle-joint *a*, which is kept in place by a counterpoise at *b*. Leather is used for packing the joints water-tight, and the inside of the nozzle is fitted with three rifle-plates, which force the water to issue in a straight line, and prevent its acquiring that rotary motion which is so common when the velocity is high, and which causes the stream to break up and lose much of its effectiveness. A 6-in. "little giant" nozzle has been worked with a pressure of 435 ft.

The discharge of water through these nozzles will be 1000 to 1500 cub. ft. per second, according to the pressure and to the size of the nozzle, the velocity reaching 150 ft. per second, and being such as can be compared only with the force of artillery. The size of the nozzle and the pressure of the water must be regulated in each case according to the circumstances, the amount of water required varying greatly with the situation and character of the bank, and the size and grade of the sluices. The supply can be regulated from the distributor. It will generally be found best to arrange for a maximum supply and pressure; in low-grade sluices, as much as $1\frac{1}{2}$ cub. yd. per miners' "inch" may be washed, while in high-grade sluices, it may reach 5 cub. yd. With a fall of $\frac{1}{2}$ in. per ft., 7 cub. ft. of water will be needed to work 1 cub. ft. of gravel.

The following directions for setting the giant are furnished by Hoskin. "In setting giants, secure as thoroughly as is thought necessary ; it is always better to stake against the bed piece and brace against the stakes, than to brace directly against the bed piece. There should be a hole in each end of the bed piece, and an iron bar driven through it into the ground. Lubricate often both joints, and under the nut of main bolt of giant ; also the ball of the deflector, which can be reached through the nozzle. Tallow is better than oil, as the latter is too thin ; when the water is very cold, axle grease perhaps is best, as it will not crumble."

Tunnels and shafts.—The next point to be considered is the location and construction of the tunnel and shafts through which the gold-bearing gravels have to be washed. The line of the tunnel will have been chosen during the preliminary survey of the ground, and with the object of meeting the following requirements :—(1) It must be perfectly straight, so as to be as short and simple as possible ; (2) it should be driven so as to reach the lowest possible point of the bed-rock that is to be worked ; (3) it should be taken as near as possible to the centre of the basin containing the gravels, before any attempt is made to connect it with the surface ; (4) it must be provided with efficient ventilation ; and (5) the grade, generally 4 to 7 per cent., must be consistent throughout, and it is better to have too steep a grade than too low, because the former may be remedied while the latter cannot.

The tunnel forms the outlet of the workings, and in it (as well as beyond it) are placed the sluice-boxes for catching the gold. The dimensions of the tunnel will therefore depend on those of the sluices, allowing about 2 to 2½ ft. of extra room ; and the size of the sluices will vary in proportion to the quantity of material and water to be passed through them, and the duration of the mining season. For single sluices about 6 ft. square, running 8 or 9 months in the year, a tunnel 8 ft. high and 7 ft. wide is commonly made, which at 4 per cent. grade will consume 2000 to 2500 miners' "inches" of water. For a consumption of 3000 to 3500 miners' "inches" of water, larger tunnels and double sets of sluices are preferable. The grade should be as high as can safely be used, both for the purpose of breaking up the material thoroughly, and to ensure against choking, at the same time that it must not be so steep as to create undue wear and tear of the sluices ; it will vary, in inverse proportion to the size of the sluice, from 4 to 7 per cent., commonly the former, and with very light and friable dirt even 3½ per cent. may suffice. In length, tunnels vary according to circumstances from hundreds to thousands of feet ; and the depth between their outlet and where they abut on the shaft bottom should be 50 to 70 ft., it being seldom easy to get more. For ventilation, an air-shaft or air-boxes

must be provided, and when the tunnel is of great length, it would be good policy to have both.

When the tunnel has been run as far as possible towards the centre of the basin, a shaft may be opened in connection with it ; but care should always be taken to avoid opening on the edge of the basin, or near the rim-rock, on account of the likelihood of meeting with quicksands and poor ground, and because, while the expenses and difficulties will be nearly the same in both cases, ground opened in the middle of the deposit will last for a long time, whereas that on the edge will be quickly exhausted. Considerable judgment is required in making this connection. The first precaution necessary is to run a drill down, in order to test the ground and gain an idea of its character. If hard, no difficulty need be apprehended ; but if soft, and still worse, if quicksandy, the greatest caution must be observed in conducting the workings, and it may even be wise to abandon the proposed shaft, and seek a new place for an opening. It is preferable, where possible, to make the connection by means of a vertical shaft ; but where the ground is soft, it is sometimes necessary to drive the bottom portion of the shaft on an incline ; and when very wet, the end of the slope must be protected, and provision made for the water to drain off without disturbing the solid materials. When the shaft drains itself, there will be little trouble in getting through sand or gravel ; but failing this, pumping may have to be resorted to, or a drainage-outlet to the tunnel may be made by putting down a bore-hole. Permanence and security must be sought in the construction of the shaft, and it must therefore be heavily and strongly timbered, and lined inside in such a way as to protect the timbers, more especially at the bottom, where very stout material must be used. The following table will give some idea of the scale of the undertaking :—

Name.	Length of Tunnel.	Average Grade of Tunnel.		Cost (reported)
		In. per Sluice Box.	Ft. per 100	
	ft.			£
North Bloomfield ..	8000	6½ in. per 12 ft.	4½	100,000
American	3900	10½ " " 14 "	6½	28,040
French Corral ..	3500	8 " " 14 "	4½	33,000
Bedrock	2600	9 " " 14 "	5½	..
Fariell	2200	6 " " 14 "	3½	..
Sweetland Creek ..	2200	8 " " 14 "	4½	18,000
Manzanita	1740	7 " " 14 "	4½	12,000
Boston	1600	10½ " " 12 "	7½	8,000
English mine	1400	12 " " 14 "	7	..

When once the connection between the tunnel and the shaft is established, such modifications as appear advisable may be readily accomplished. An incline may be turned into a shaft, or a shaft into a

series of terraces. The latter are considered by some engineers to be more effective in breaking up cement and firm gravel than a single deep drop ; when used, they should be arranged so that the fall decreases as they descend :—thus, with 100 ft. fall, the first may be 30 ft., the second 25 ft., the third 20 ft., the fourth 15 ft., the fifth $7\frac{1}{2}$ ft., and the last, into the sluice, $2\frac{1}{2}$ ft. It now and again happens that the lay of the ground and the thickness of the deposit (say 200 to 300 ft. in the centre) may render it possible to commence operations without the aid of a tunnel, by arranging ditches so that the gravel is being worked and returns made while the driving of the tunnel is proceeding. The shaft should always have a safe, water-tight, and well-ventilated compartment, by which the workmen can ascend and descend, for the tunnel must not terminate at the bottom of the shaft, but be prosecuted through to the farther rim-rock, connection between the inner and outer portions of the tunnel being suspended meantime.

The following remarks on tunnels are made by Russell Dunn, in the California State Mineralogist's Report for 1888 :—

“As to the dimensions to be given to the tunnel there is great diversity of opinion among miners, and the practice varies correspondingly. The dimensions of the older tunnels approximate in clear width 5 ft. on the bottom and 3 ft. at the top, with a clear height of 6 ft. Some of the tunnels of the recently opened mines have dimensions of 8 to 10 ft. width at the bottom, and 7 to 8 ft. clear height, the top being arched in untimbered rock, or having a clear width of $3\frac{1}{2}$ to 4 ft. in timbered. The convenience of the larger tunnels over the smaller is undeniable, and results in larger aggregate savings in the time of the men employed in the mine when going to and from work, and in a measure protects them from the liability of petty accidents. While this is conceded, the point at issue is the relative initial cost. This cost is made up of the cost of breaking down the rock, which includes drilling and powder, and the cost of taking the excavated rock out of the tunnel. In soft rock, timbering is also an item. Experience has demonstrated that greater advantage can be taken of the rock in drilling the blasting holes, and that it does not take any greater number of inches of holes drilled for a large tunnel than for a small one per foot of length, and that it takes, if anything, but little more powder. Holes can be drilled deeper and a breaking angle obtained in a large tunnel, so that the full effect of the powder is obtained. In a small tunnel it is difficult to get a breaking angle, and a deep hole is more liable to blow out. The cost of breaking the rock is often actually less for the large tunnel than for the small one. The moving of the rock may cost more in a large tunnel, as the quantity is so much greater. For a tunnel 6 ft. high, 5 ft. bottom, 3 ft. top, the rock in place per running foot measures 27 cub. ft. ; broken down, about

31 cub. ft. ; in a tunnel 8 by 8 ft., the top arched, the amount is 60 cub. ft. in place, 68 cub. ft. when broken down ; and in a tunnel 8 ft. high, 10 ft. bottom, top arched, the figures are, respectively, 75 cub. ft. and 85 cub. ft. In practice, if hand drilling be employed, one carman can remove the rock from a long tunnel as fast as it can be broken down, even if it have the extreme dimensions noted. If machine drills are used, and the ground be easily broken, a second carman may be necessary, and a longer time is needed to get the broken rock from the face, but this is far more than compensated for in the larger tunnels by the opportunity to work two drills at once.

" The difference in the cost of timbering between the large tunnel and the small is very little, if anything at all. Timbers cost by the piece, the price depending more on the relative scarcity of suitable trees and the distance they have to be hauled, than on their length and size. If lagging is used, the larger tunnel requires 20 per cent. more than the smaller. As a general proposition it may be safely taken that the large tunnel will not cost per running foot to exceed 10 per cent. more than the small one, and that to offset this there will be at least 20 per cent. gain in the rapidity of the work with hand drilling and $33\frac{1}{3}$ per cent. with power drilling.

" The speed with which a tunnel can be made and its cost are almost entirely dependent on the kind of rock penetrated. In serpentine, by double hand drilling, timbering being necessary by reason of the scamy and blocky character of the rock, which has little coherence, the cost will average \$4 a foot for a tunnel 8 ft. wide on bottom, $3\frac{1}{4}$ ft. on top, and 7 ft. high in the clear. Of the \$4 (the labour costing \$2.50 a day), \$2.85 is for labour, including one carman and blacksmithing, 65 cents for superintendence, 30 cents for powder, and 20 cents for timbers and lagging. A 30 per cent. nitro-glycerine powder will produce the best results in blasting. A nine hours' shift of four miners, one carman, and blacksmith working half time, will excavate and timber between 4 and 5 ft. of tunnel. It would not be advisable to use machine drills in this kind of rock. In hard, compact, metamorphic limestone and in granite only, a much less rate of progress is possible. By hand drilling in the limestone, the rate of progress, no timbering being necessary, would not average over 1 ft. per shift, the drilling being very difficult, and the rock requiring a higher grade of powder and more of it. The cost per running foot would be from \$15 to \$22. In granite, somewhat better results are possible, the rock both drilling and blasting better. The cost per foot would be from \$10 to \$18, dependent on the variations of hardness in this rock. Machine drilling in the limestone is more expensive, but more rapid than hand ; the cost would go as high as \$25 a foot ; the rate of progress to $2\frac{1}{2}$ ft. a shift. In granite, the cost of the tunnel by

machine drilling would be about the same as by hand, but progress would be twice as rapid, as 4 ft. a shift could be made. The rocks already noted, serpentine, limestone, and granite, are exceptional, the common bedrock being slate in every possible degree of hardness, from the soft talcose and almost decomposed strata, to hard, compact, blue slates, resembling the traps and greenstones. Through these, cutting the stratification at every angle, have been run nine-tenths of all of the drift mining tunnels.

"The greatest variations in the cost per foot exist, together with equally wide divergences in the rapidity of excavation. Some tunnels have been run for \$3 a foot, others have cost at the rate of \$75. The estimate of probable cost of any proposed tunnel is very largely speculative, as there is no uniformity of quality in the different strata that one tunnel line may go through, nor even in the same stratum.

"The figures of tunnels that have been run, make it quite safe to estimate the probable cost of a long tunnel, 8 by 7 ft. in slate, using power drills, at \$14 or \$15 a foot for the cutting alone. The timbers and lagging, if used, will cost 15 cents to 27 cents a foot, the expense of setting them up being more than offset by the greater penetrability of the rock where it is necessary to use them. The track, preferably T rail weighing 30 lb. to the yard, will cost 45 cents to 65 cents a foot laid. The power plant, including 3½-inch drills (3-inch will do for soft rock very well), compressor for water or steam power, as circumstances may determine, piping and blower for ventilation, will cost, set up in place, from \$3000 to \$6000, dependent largely on the accessibility of the tunnel to the transportation facilities."

Sluices, drops, grizzlies, under-currents, &c.—The tunnel completed and connected with the shaft, the next object of attention will be the sluices. Already much has been said in preceding chapters, pp. 169–80, 193, 287, concerning sluices; but though the principles of their construction and function remain the same in all cases, the sluices for hydraulic mining require to be of a character in proportion to the increased work expected of them. The sluice-boxes are laid throughout and beyond the tunnel to the point where the tailings are to be deposited or "dumped." The chief objects of attention in laying them out are their grade, and to make them at once permanently strong yet easily movable.

The first set of sluice-boxes will be placed in the tunnel, being either single or double, according to the capacity of the tunnel, and arranged in such a way as to take the largest possible quantity of gravel. When it is intended to wash very large quantities of gravel, the sluices are made double, resting on the same sleepers, and parted by water-gates so placed that the two sections may be worked simultaneously, or the full stream be turned into one partition to the exclusion of the other. These

gates are useful also in allowing one part of the sluice to be cleaned up while the other is working, and they greatly facilitate repairs for the same reason. Windlasses and ropes are required to move them when the sluice is full of water. The sluice is sometimes made double only in that section where repairs are anticipated.

The sluices outside the tunnel will vary in length, according to the nature of the dirt being washed, and must be determined in each instance by actual experiment, an increase being necessary if assays of the tailings show a loss of gold. The sluice is generally run on a wide curve, the outside edge being raised $\frac{1}{2}$ to 1 in. to equalise the wear and tear, because in a straight sluice the velocity of the current would be likely to carry away much of the dirt without disintegrating it. The dimensions of the sluice are determined by the quantity of material to be treated, which is governed by the water supply. One 6 ft. wide by 3 ft. deep, with a 4 to 5 per cent. grade, will take about 3500 miners' "inches;" one 4 ft. wide by $2\frac{1}{2}$ ft. deep, with $2\frac{1}{2}$ per cent. grade, 1200 to 1500; or with 4 per cent. grade, 2000 miners' "inches." The water must be in sufficient depth to cover the largest boulder likely to be encountered, so that the body required will vary with the coarseness or fineness of the dirt. With too much water, the riffles are likely to pack, and the yield of gold will be less, increasing with low grade and small body of water. If water is plentiful and cheap, it will to a certain extent atone for low grade; but with water scarce and dear, a high grade is essential. Generally speaking, ordinary gravel needs 4 per cent. and coarse gravel 6 to 7 per cent.: the heavier the gravel the steeper the grade and the more water necessary; 4 per cent. is very commonly used, increased to 6 and even 8 for clay, and reduced sometimes to $1\frac{1}{2}$ per cent. for very light dirt.

Water consumed.—At the "No. 8" mine of the North Bloomfield Gravel-Mining Co., during the time from Jan. 1, 1875, to Oct. 3, 1877, 7,071,630 cub. yd. of gravel were washed, with an expenditure of 3,750,797,560 cub. ft. of water. This gives an average of 534 cub. ft. of water required to wash 1 cub. yd. of gravel; or, in other words, the gravel at this locality required for moving it an expenditure of water nearly equal to 20 times its bulk. At the Blue Tent Co.'s mine, where careful record has been kept of the amount of gravel washed, water used, &c., for the past few years, the various kinds of gravel met with were moved at the rate of from 2·38 to 10·12 cub. yd. per miners' "24-hour inch" (which is considered on the San Juan Divide equal to 2230 cub. ft. of water; in the Bear River mines, about 2200), or, in other words, the gravel required, according to its condition, from 8 to 34 times its volume of water to disintegrate it and carry it into the sluices. That which demanded the largest quantity of water specified is described as being "hard, indurated, and clayey."

Hague adopted 7 cub. yd. as the amount of gravel which, on the average, in the divide between the South and the Middle Yuba, could be moved by a "24-hour inch" of water, and this is said by Professor Pettee to be corroborated by the results obtained at Smartsville; 7 cub. yd. to the "24-hour inch" gives an amount of gravel not quite $\frac{1}{12}$ of the volume of the water used. Ashburner considered that a "24-hour inch" of water would move only about $3\frac{1}{2}$ cub. yd. of the lower portion of the gravel deposit in Bear River and its tributaries. This gravel may be considered as representing the hardest kind ordinarily worked by the hydraulic method.

It appears, therefore, that a "24-hour inch" of water will disintegrate and carry into the sluices 2 to 10 cub. yd. of gravel, according to the character of the material. This, with the data previously given, will give an idea of the amount of water required in hydraulic operations.

When suspending washing operations, it will still be advisable to keep a stream of water flowing through the sluice, to prevent the joints opening, and a consequent loss of mercury and amalgam on resuming work.

Erecting sluice.—After laying out the line and grade of the sluice outside the tunnel, sills measuring 4 in. by 6 in. at least, and 15 ft. long for a double sluice, are placed on the ground 4 ft. apart. Posts 4 to 6 in. sq. and 3 to $3\frac{1}{2}$ ft. long are then fixed, and braced together with $1\frac{1}{2}$ -in. planks; a single 8-in. plank is run from sill to sill on each side to serve as a foot-board, and facilitate and strengthen the construction. The bottom and sides are of $1\frac{1}{2}$ -in. planking, battens being nailed outside over the joints to close the interstices. Next, the lining and paving are introduced.

Paving.—The paving is of various descriptions. In Fig. 107 are seen two sections of a sluice paved with stones. These are oval, placed edge-wise, and set with a slant in the direction of the current, their thickness being usually 9 to 12 in. No stones should be used of less than 20 lb. in weight, or the power of the water will displace them. When quarried rock is used, it is laid in the box with a thickness of 15 to 18 in. One man can lay in a day 8 boxes 12 ft. long, with stone 15 in. deep, the width of the box being not less than 3 ft. nor more than 4 ft. The paving is done in compartments, each 6 to 8 ft. long, and held in place by pieces of stout planking fixed across the sluice. This precaution prevents the pavement getting loose, and facilitates repairs when needed. The larger spaces between the stones are filled by running through waste gravel, while the smaller interstices on the top serve the purpose of riffles. Paving of this kind will weigh about 8 tons per every 10 ft. of a double sluice, necessitating a strong structure. A side lining *a*, of 2-in. plank or worn wooden blocks, is nailed on the inside of the sluice, projecting about 2 in. below the surface of the stones, and rising about



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**PIPING, OR HYDRAULICING,
NORTH BLOOMFIELD HYDRAULIC MINE, NEVADA COUNTY, CAL.**

10 to 15 in. up the sides ; the wear on this is partly saved by an additional lining *b*, 6 to 8 in. wide ; *c* is the boarding of the sluice itself ; *d*, the footboard on either side. A run with a rock bottom may last for 100 to 150 days.

Another mode of paving is with pine blocks, 12 to 18 in. sq., cut across the grain of the wood, and 10 to 12 in. high. They are arranged as shown in Fig. 108, secured by a plank $1\frac{1}{2}$ in. thick by 6 in. high, nailed to the lower part of the block with headless nails, which stand up about 1 in., the succeeding blocks being driven down on the projecting nails so that they come close to the cross-piece. When the sluice has thus been filled with blocks, the side lining *a* is introduced. The spaces between the rows of blocks correspond with the $1\frac{1}{2}$ in. cross-pieces. The wood

FIG. 108

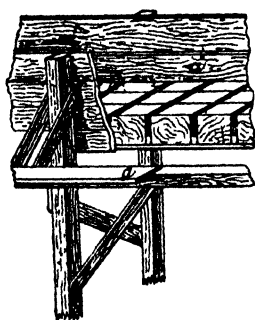
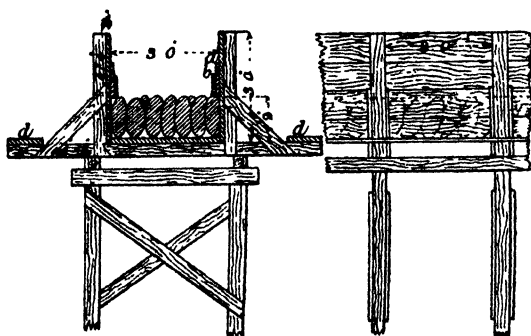


FIG. 107.



PAVING FOR SLUICES.

preferred at Smartsville is that of the "digger" pine (*Pinus Sabiniana*) and the blocks are cut 7 in. thick. A run lasts 20 to 30 days, when the bottom becomes so uneven that repairs and a clean-up are necessary.

Sometimes the heaviness and character of the wash-dirt compel the use of riffles laid longitudinally, instead of horizontally. The best are old railway-rails, when procurable ; but a moderately good substitute may be made by covering wooden rails with strap-iron. Iron rails are laid close together, easily placed and easily retained in place, and give excellent results.

With regard to the relative claims of these several kinds of riffles :—Wood is expensive to build, presents the least catching surface, and lasts but little more than a month, the rapid wear rendering it likely that the gold may be ground out and washed away. Stone is the cheapest material, but expensive to pack in and remove ; it wears slowly, and may be most conveniently used outside the tunnel, where room for piling the stones is at hand, and it presents the greatest opportunities for catching the gold, so that on the whole it is probably superior to wood. Iron rails

have perhaps given the best results of any, and are becoming very popular.

Under-currents.—In setting out the sluice, provision will have been made for a sufficient number of under-currents. These, as shown in Fig. 109, may be likened to shallow ponds *c*, varying in form from oblongs or triangles to no regular shape, and in size from 10 to 40 ft. wide and 30 to 100 ft. long. They are put outside the tunnel, as circumstances of the ground will permit, about 75 to 100 ft. apart, the main object being to secure a space of some 500 to 1500 sq. ft., without much regard to its exact shape. The duty of the under-current has been already explained on p. 180. The sand and water are conducted into it by boxes 15 to 18 in. wide, placed in the bottom of the sluice, the outlet being fitted with a grating of 1-in. square steel bars set 1 in. apart. The gold-bearing muddy waters thus rush into the under-current at considerable velocity, and spread themselves out over its broad surface, which is furnished with riffles of different kinds, whose effect is to check the current, and cause the deposition of the gold and heavy matters, the former being caught by the amalgam and mercury placed there for the purpose. The riffles are arranged in such a way that the removal of a few slabs enables all or any part of them to be taken up. The grades of under-currents vary from 10 to 17 in. per 12 ft. Being very wide, check-boards are set at the upper end, to distribute the stream as evenly as possible over the whole floor; and as a considerable quantity of water is thus withdrawn from the sluice, its width is commonly reduced by 6 to 12 in. for the length of the under-current, resuming the full width again beyond. The lay of the ground and the amount of grade will determine whether the under-current shall be all on one side, on alternate sides, or on both sides at once of the sluice; in the last mentioned-case, the sluice must be narrowed twice as much.

The discharge of the sluice and the under-current is usually made into a drop-box *a* (Fig. 109), which is much wider than the sluice, built of stout timber to withstand rough usage, and covered at bottom with a heavy stone paving, 4 or 5 ft. below the level of the sluice, and deeper than the continuation of the sluice, so as to serve as a receptacle for heavy matters. The depth of the drop is often ruled by the length of the under-current, which generally discharges into it over the side; but occasionally it is governed by the lay of the ground, so that the under-current will discharge from an end sluice. When a fall of 40 to 50 ft. is possible, the drop is placed directly under the sluice, which is continued either from one side at right-angles to its direction above, or, when the ground allows, in the same line, the continuation of the sluice being merely a contraction of the drop-box.

Grizzlies.—The “grizzly” *b* (Fig. 109) is a projection of the main

sluice beyond and over the drop-box, and is an admirably simple contrivance for getting rid of the boulders and stones, which contain no gold, require much water, and only wear out the sluice. It can be availed of only when there is abundant natural accommodation for the *débris* thus ejected. The projection of the sluice over the drop-box is floored with railway-rails set 6 in. apart, and is inclined at an angle of 25° to 30° , to give the large bodies sufficient impetus to ensure their passing over; while everything small enough to go between the rails finds its way into the drop-box.

General arrangement.—The under-currents, drops and grizzlies are repeated as often as circumstances permit or require. At one works, there were 10 miles of sluice 4 to 6 ft. wide, and 23 under-currents 10 to 40 ft. wide. A sluice intended for running off and rewashing the tailings that had been accumulated for 20 years was 6 miles long, 16 ft. wide, and 12 ft. high, built of heavy lumber, and anchored to the bed-rock with iron bolts. It was divided into two compartments, one 10 ft., the other 6 ft. wide, both being used when water was abundant, only one when it was scarce, or the other was being cleaned up.

On the North Bloomfield Company's workings, the arrangement of the sluices and under-currents is as shown in Fig. 110. A sluice paved with rock starts from the mouth of the tunnel *a*, and thence a variety of cuts and sluices conducts the current over several under-currents set on different grades, paved in turn with rock, blocks, and longitudinal riffles covered with strap-iron. The grizzlies used are made of 4-in. $\times$ 1-in. wrought iron, set on edge. The discharge from the under-currents is taken up by the main sluice and subsequently redischarged over the succeeding under-currents, till the last sluice and under-current deposit the tailings finally in a cañon. The under-currents are as follows:—

1. Size, 24 $\times$ 36 ft.; grade, 13 in. per 12 ft.; chute, 2 ft. wide at opening, contracted to 10 in.; iron rail riffles. Yield of amalgam of the various compartments:—*b*, 108½ oz.; *c*, 83¾; *d*, 46½; *e*, 31¼; chute, 46¼; total, 316¼ oz. in three clean-ups.

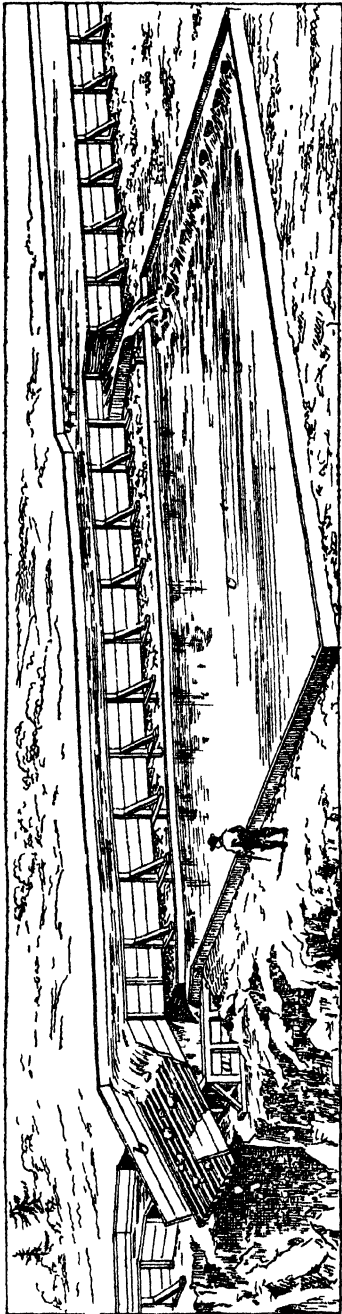
2. Size, 24 ft. $\times$ 24 ft.; grade, 12 in. per 12 ft.; chute, 2½ ft. wide at opening, contracted to 2 ft.; iron rail riffles. Yield:—*b*, 48¾; *c*, 36¼; *d*, 20¾; *e*, 23½; chute, 14; total, 143¼ oz. in two clean-ups.

3. Size, 24 ft. $\times$ 36 ft.; grade, 15 in. per 12 ft.; chute, 2½ ft. wide at opening, contracted to 2 ft.; rock riffles. Yield:—*b*, 50½; *c*, 35¼; *d*, 18½; *e*, 16; chute, 8½; total, 128¾ oz. in two clean-ups.

4. Size, 20 ft. $\times$ 36 ft.; grade, 12 in. per 12 ft.; rock riffles. Yield, 71¾ oz.

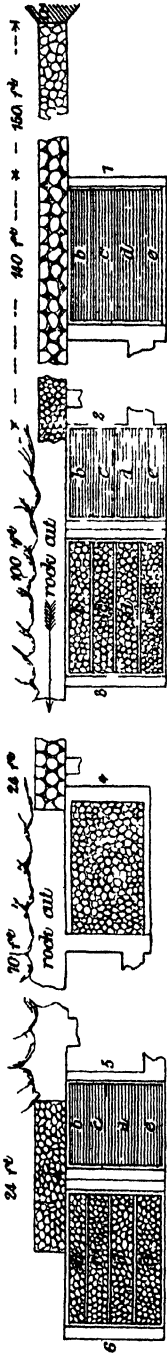
5. Size, 24 ft. $\times$ 24 ft.; grade, 12 in. per 12 ft.; chute, 2½ ft. wide at opening, contracted to 2 ft.; riffles, 4-in. $\times$ 1¼ in. lumber covered with strap-iron and placed 1 in. apart. Yield:—*b*, 5; *c*, 8½; *d*, 5; *e*, 6½; total 25 oz. in one clean-up.

Fig. '09.



HYDRAULIC UNDER-CURRENT.

FIG 110



GENERAL ARRANGEMENT OF SLUICE, DROPS, GRIZZLIES, AND UNDER-CURRENTS.

6. Size, 24 ft. $\times$ 36 ft. ; grade, 17 in. per 12 ft. ; chute, $2\frac{1}{2}$ ft. wide at opening, contracted to 2 ft. ; rock riffles. Yield :—*b*, 8 ; *c*, 5 ; *d*, $3\frac{1}{4}$; *e*, 3 ; total, $19\frac{1}{4}$ oz. in one clean-up.

The water used was 350,000 miners' "inches" of 24 hours each ; the total yield in 6 months was about 700*l.* worth of gold, of which about 630*l.* were taken from the first 150 ft. of the sluice down to the first under-current, and the balance from the remainder of the sluice.

Tail-sluices.—Of the immense mass of material washed into the sluice, but a small portion only is removed by the grizzlies, and all the remainder has to be disposed of. For this purpose, "tail-" sluices are run, if possible, to a large river which will be capable of carrying away the tailings, or failing that to a smaller stream or a ravine. The tail-sluice must be stoutly and securely built, at the same time that it is easily taken down and transported as the exigencies of the case may require. Nothing needs more careful attention than the tail-sluice, for the body of barren material which has to be got rid of is so enormous, that rivers 500 ft. wide, 14 ft. deep in mid-channel, and with a fall of 18 ft. per mile, have been quite filled in 11 months of hydraulic washing ; and it must be borne in mind that the moment it is impossible to get rid of the tailings, the works must be shut up.

Blasting.—The removal of the bank of gravel which is to be washed should be effected as far as possible by the aid of water alone ; but under certain circumstances, water may not suffice for the operation, as, for instance, when the gravel is very hard or cemented, or the bank so high that the nozzle cannot safely be put within close enough range, or for the purpose of accelerating the action of the water. In such cases, recourse is had to blasting, with the object of shaking and loosening the earth.

There are two ways of effecting the blast, either by a shaft or by drifts. In the former case, the shaft is either enlarged somewhat at the bottom, or a few little drifts are run from it ; the effects of such blasts being necessarily restricted to a small area, they are fit only for limited operations. Such shafts are occasionally used to loosen the upper gravel when the bottom stratum is a tough cement. In this way, the top is worked off first, and the bottom afterwards blasted by cross-drifts. They are usually 4 to 5 ft. in diameter, and are tamped with the material excavated. In the latter case, a number of drifts are run in proportion to the extent of ground to be blasted—which may be of almost any desired size. Egleston recommends that the main drift should be as small as it can well be worked, say 4 ft. high by 3 ft. wide, of the same length as the bank is high, and crossed by one or more auxiliary drifts, according to the size of the blast, one placed across the end and about half as long as the main drift, the second (if only two) across the middle and about one-third as long as the main. For a bank 80 to

120 ft. high, about 600 kegs of powder of 25 lb. each will be needed, of which there should be distributed two-thirds in the cross drifts at the end of the main drift, and one-third in the half-way cross-drift. A bank of 80 to 85 ft. will take 400 to 500 kegs ; one 250 ft. high, 1500 to 2000 kegs. Bowie says that the main drift should be run for a length equal to two-thirds the height of the bank, and cross-drifts from the end of the main drift should be carried parallel with the face of the bank, the length being determined by the extent of ground to be blasted. A single T is all that is necessary. The powder needed will vary according to the character of the bank and gravel from $\frac{1}{2}$ to $\frac{3}{4}$ keg (minimum) per 1000 cub. ft. of ground, which may be estimated as the height of the bank $\times$ the length of the main drift $\times$ the length of the cross drifts. The quantity of powder used should in any case be thoroughly sufficient to do the work, and had better be in excess than too little, as a blast inefficiently performed will not repay the cost and trouble of cutting the drifts.

Usually the powder is emptied out of the kegs into long boxes, which are then provided with fuses at intervals corresponding to about 50 kegs, this plan tending to secure the complete and simultaneous combustion of the powder. When there is water in the drifts, these boxes must be well tarred, and the holes for the fuses should be made in them and closed and coated with wax. Sometimes the heads are simply knocked out of the kegs and a fuse is placed in each, but the uncertainties of simultaneous combustion are then increased, by reason of the greater number of fuses necessary. When the fuses have been fixed, wires connecting with the battery are laid in place, and the main drift is securely closed at the first cross-drift by means of stout timbers, and then filled up to its mouth with damped sand. The fuses are fired by an electric battery. A variety of explosives have been tried for blasting, but powder is the general favourite, on account of safety and ease of transportation.

Of Judson powder, an explosive that is coming into extensive use for bank blasting, a much less quantity suffices than of ordinary black powder. Judson powder is, by many experienced miners, preferred to every other, being recommended both for its greater economy and efficiency. Giant powder has occasionally been used for bank blasting with good effect. Where it is employed, each separate lot must be closely tamped, so as to wholly exclude the air. Though much more expensive, 1 lb. of this powder will do as much execution as 5 or 6 lb. of ordinary powder. In charging the drifts, the kegs or boxes of powder are placed along them in rows. The tops of the middle row of packages, whether kegs or boxes, are then removed, and two wires are laid along them and connected with others that extend to the electric battery with which the explosion is effected. From 8 to 12 electric fuses are equally distributed and buried in the powder. Where Judson powder is used,

the exploders are inserted in giant powder caps and laid on the top of the paper covering by which this explosive is protected from moisture.

Giant powder is much used for breaking up the boulders found in hydraulic banks, so that the fragments will pass through the sluices. The cartridges are broken open and the powder is poured on to the rock, and sometimes built over with clay to hold it in place or increase the effect of the blast, a fuse and cap being used to explode the charge. Boulders which could not be reduced by sledge hammers, nor blasted by black powder without drilling, are thus easily broken down to such sizes that they will be carried away in the sluice. Pipe-clay also is now overcome by blasting with giant powder. The lumps of clay rolled down from the face of the washing are bored to the centre by a clay auger, and charged with $\frac{1}{4}$ or more of a cartridge of giant powder, with fuse attached. The shots are fired by a hot-iron and rod. This disintegrates the masses so that the water can completely reduce them.

It has been very clearly pointed out, in a recent report of the State Mineralogist of California, that in bank blasting, the earth which has been upheaved and shattered is ordinarily worked into the flumes at the earliest practicable moment, sufficient time not being allowed for it to be acted upon by the combined influences of water and air. Minute particles of gold are often held in physical contact with larger bodies of some other material, the specific gravity of which may not be more than $2\frac{1}{2}$ or $3\frac{1}{2}$. It is washed into the flume before disintegration can take place; in a few minutes, or an hour, it is in the dump, where it undergoes the process it should have been subjected to before entering the flume. The non-disintegration of a considerable portion of a hard gravel bank, or one that requires blasting, is the cause of one great loss that is known to be sustained in hydraulic mining. Hence it has been suggested that the magnitude of blasts should be increased enormously; so as to break up ground 2 or 3 years before it will be sluiced. "The great mass thus shattered and broken, would be the contents of a huge leach tub with a filter at the bottom. The lower and more valuable strata have remained for thousands of years with absolutely no change of position or temperature. The admission of running water from time to time upon the shattered, but hitherto motionless and heavily compressed deposits, will impart its caloric to the mass, thus inducing motion in its particles, a process of disintegration. And, if sufficient time be given it, with alternate exposure to air and water, the material that now passes through the flumes holding gold captive, will set it free before it enters the flumes. Then, the blasting can and should be done years in advance of the piping. It will change all the material into the best possible condition for working."

The many advantages of electric blasting have forced themselves upon miners, and now the managers of few well conducted mines are

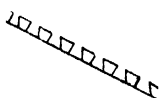
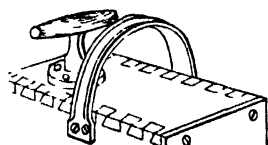
content to rely on the older and less effective plans. Those who have most carefully studied the matter are most earnest in praise of the method and its economical value. A very little thought will make apparent the greater effect which can be produced by firing simultaneously a number of blasts instead of firing them singly, while a little experience will teach that even in firing single blasts by this apparatus, much can be gained. One advantage gained in firing single holes is that in case of missfire, which can rarely happen by this method, no time is lost in waiting, as in the case of firing by safety fuse there would be, before approaching the work. There is no "hanging" fire. Another advantage is that the explosion of the electrical fuse in the centre of the charge throws the fire through the whole body of the powder, igniting it all at once and by detonation, giving the same charge by far greater explosive effect, as has been fully demonstrated by experiment.

Following are directions for operating. Use a fuse with the wires attached, of such length that the ends may protrude from the surface after the hole is charged, the fuse head

FIG. 112.

being in the centre of the charge. Tamp with dry sand, or in such a manner that the wires may not be cut or the insulating covering upon them be injured. When all the holes to be fired at one time are tamped, separate the ends of the two wires in each hole, joining one wire of the first hole with one of the second, the other, or free wire, of the second with one of the third, so proceeding to the end or last hole. If the wires attached to the fuses should not be long enough, use connecting wire for joining. All connections of wire should be by hooking and twisting together the bare and clean ends, and it will be best if the parts joining be bright. The charges having all been connected as directed, the free wire of the first hole should be joined to one of the "leading" wires, and the free wire of the last hole with the other of the two leading wires. The leading

FIG. 111.



ELECTRIC BATTERY AND
FUSE FOR BLASTING.

wires should be long enough to reach a point at a safe distance from the blast, say 250 ft. at least. All being ready, and not until the men

are at a safe distance, connect the leading wires, one to each of the projecting screws on the front side of the machine, through each of which a hole is bored for the purpose, and bring the nuts down firmly upon the wires. Now, to fire, taking hold of the handle of the magneto-electric battery, shown in Fig. 111, lift the rack (or square rod toothed upon one side) to its full length, and press it down, for the first inch of its stroke with moderate speed, but finishing the stroke with all force, bringing the rack to the bottom of the box with a solid thud, and the blast will be made.

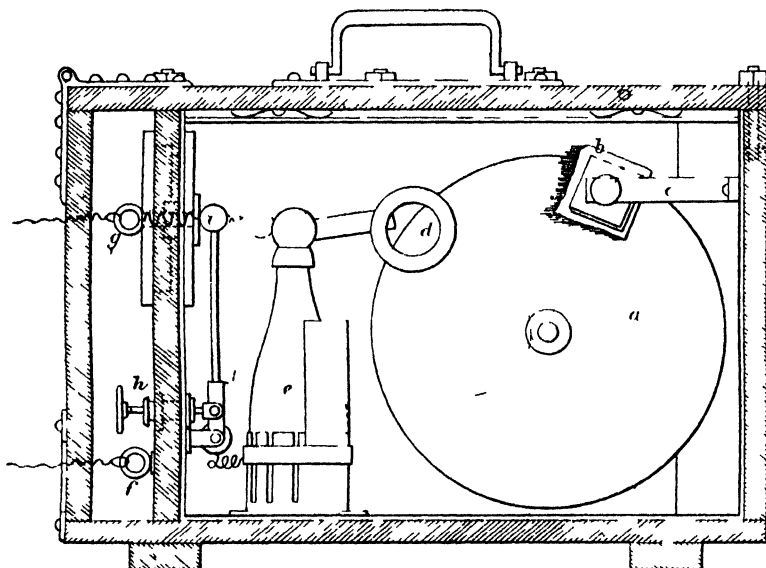
Fig. 112 shows the fuses which are to be used with the magneto machines. The cut shows in section a platinum fuse nearly of actual size. These fuses are kept in stock with wires of 4, 6, and 8 ft. in length. Fuses with longer wires are made to order. The length of wires should be equal to depth of holes drilled. Enough leading wire is needed with each machine to make two leaders of sufficient length to reach from the blast to a safe distance for the person to stand who shall operate the machine: 500 ft. is the quantity usually sold, but in some cases 1000 ft. are used. Connecting wire is sold in coils of 1 or 2 lb. weight (about 225 ft. in each coil), and is used in connecting the fuses to each other where several charges are to be fired simultaneously. A small quantity only of this will be needed if the fuse wires are picked up after each blast, as they can be twisted together and used in place of connecting wire.

The apparatus described and illustrated above is furnished by the Ingersoll Rock Drill Co., 10, Park Place, New York. The price of their magneto machine, to blast 12 holes, costs about 5*l*.; the platinum fuses range from about 16*s*. a hundred with 4-ft. wires, to 2*l* with 16-ft. wires; leading wire, $\frac{1}{2}$ *l*. a foot; connecting wire 2*s*. a lb.

Bornhardt's electrical firing machine is designed to excite electricity by means of friction, the electricity so excited being stored in a receiver known as a Leyden jar, whence it can be discharged suddenly by pressing a knob. The electricity thus discharged is of very great power, sufficient to leap over, in the form of a vivid spark, any small breaks which may occur in the circuit or path taken by the fluid. It is this property which is taken advantage of to fire the detonator. Small wires are led into a specially prepared detonating powder contained in the fuses. On connection being made with the firing machine, the current passes along the wires, leaps across the break, which occurs in the circuit in the form of a spark; this spark explodes the fulminate, which in its turn explodes the charge of dynamite or other explosive near it. The passage of the electric spark is so rapid that it may be carried through any number of charges up to 20, all of which it will explode simultaneously. The machine (Fig. 113) is composed of a thin plate wheel *a*

of ebonite, which can be rotated rapidly by turning the handle. The top part of the wheel *a* is in contact with a rubber *b*, formed of cat's skin, carried by the arms *c*. It is the friction of this skin on the surface of the wheel which excites the electricity *d* are two ebonite rings furnished on the inside with metal points and connected with the Leyden jar *e*, in which the electricity is stored after having been collected from the surface of the generating wheel *a* by the points on the rings *d*. Metal rings *f g* project through the front edge of the machine; it is to these rings that the ends of the wires forming the circuit are to be attached when everything is prepared for firing. *h* is the firing knob by which the discharge is made. The knob being quickly pressed causes the arm *i* to be thrown into the position shown in dotted lines, making contact with the top of the Leyden jar and completing the circuit, allowing the charge stored in the jar to pass along the wires, and in its passage firing the prepared shot holes simultaneously.

FIG 113



BORNHARDT'S ELECTRICAL FIRING MACHINE

Conduct of Operations.—When everything has been made ready, the washing operations commence. The first step is to remove the upper timbers of the shaft to a depth of some feet, and to shape out the ground around the shaft into a basin, the water and dirt being all washed down the shaft and through the tunnel, and helping to fill up the spaces between the riffles, and the cracks and openings in the sluices. This

basin must be shaped as rapidly as possible, so as to get a space for the nozzles to work ; but care must be taken not to choke the shaft. The successful future working of the bank depends in a great measure upon the care bestowed on the preliminary opening. To wash down a bank, two nozzles are commonly used, and made to play at the foot of the bank, at an obtuse angle, one on either side. Sometimes a third nozzle will be needed to wash away what the other two bring down, and sometimes one does the cutting down and a second the washing away. At first the water spreads and splashes all abroad, but soon it commences to bury itself, and gradually works out in the bank an arched cave, whose depth is regulated by the judgment and experience of the miner.

When one cave is finished, a second is commenced ; and when several are made, the portions left standing between them are washed away, and

FIG. 114



BLANKET-TABLES AT END OF HYDRAULICING SLUICE

the upper part of the bank falls down. The operation is illustrated in Plate VI. The fallen material is washed into the sluice or sluices

connecting with the top of the shaft, and thence into the tunnel (if there be one, which is not the case in the example shown) and sluices for extracting the gold. At Reefton, New Zealand, it is finally run over the blanket-tables shown in Fig. 114. It is important that the supply of earth should be kept as regular as possible. The ground is worked in the shape of a funnel, whose leg is the shaft ; and sometimes the shaft itself is washed away, so that only a deep basin exists, with the tunnel for an outlet at the bottom. It is most economical to work several nozzles in the same general direction. Rubbish which is not worth putting through the sluice-boxes should be deposited where it may remain and not need removal. Boulders too large for the sluice must be broken down by hammers or powder.

An important precaution in washing a bank is to keep its face square, as it is then cheaper, easier, and safer to work. If the banks are allowed to overhang, the men's lives are in danger from a sudden fall of earth, which they could not avoid. When the banks are very high, they are best worked in terraces of about 125 to 150 ft. high each ; 200-ft. banks are sometimes washed off at once, but they are very dangerous. The bank generally gives warning of an approaching fall, and the water should then be shut off, for if it be allowed to play upon the falling mass, the dirt would run, and the men might have great difficulty in avoiding it ; whereas with proper precaution, the boulders only will run, and they can be seen and escaped. Falls, or "caves," as they are technically called, are usually effected before night, to give time for remedying accidents before the next day's work. In very large washings, work is continued at night to run through the "cave ;" but in small ones, night-work is unusual, except the pay-dirt be very rich, when, to use Egleston's own words, "the work is continued as much to prevent clean-ups [by other people] as to get the pay in a shorter time."

A method of moving very large boulders is to undermine them and let them roll. Boulders 6 ft. in diameter are sometimes moved thousands of feet in this way ; but the plan is eminently dangerous, despite the possibility of judging what direction the rock will take. It is better to use a simple derrick, mast about 100 ft. high, set in iron sill, and supported by guys, and boom about 90 ft. long. This can be easily moved without taking down, and will lift anything up to 10 or 11 tons ; larger boulders must be blasted. The power used for the derrick is generally a simple overshot water-wheel.

When commencing with new sluice-boxes, light gravel must be washed through with a full head of water for a day or two, to fill up the greater part of the spaces and hollows in the paving, otherwise the mercury when added would sink out of reach of the gold. Before beginning the washing proper, the current of water is reduced, and 500 to

600 lb. of mercury is introduced for a sluice 500 ft. in length. The bulk of this will be placed in the upper portion of the sluices, as the stream always washes it down more or less ; and it should be sprinkled so as to fall in little globules. The whole quantity is not introduced at once, but in little dribbles, till it is visible behind the riffles. Sometimes the quantity is progressively diminished during the final weeks of a "run" ; but a better plan is to carefully watch the sluices, and add whatever may appear necessary up to the last moment. The sluices are usually attended to twice a day, and a little mercury is added.

General clean-ups are not usually undertaken more than once or twice a year, when the whole operations are suspended, and the paving and riffles are removed from the sluices ; but partial clean-ups, especially in the upper part of the sluices, occur about once a fortnight. When the clean-up is to be made, the sluice is run empty, and sections of the paving are taken up in succession, commencing at the top. Riffles are then put across the sluice, a little mercury is poured in to collect the amalgam, and the fine heavy matters collected are washed down with a small stream. The amalgam is scraped up, as in any other sluice. To guard against stealing, the sluices are sometimes filled with gravel before turning off the water, and this is removed by hand, section after section, as the cleaning up progresses ; in other cases, a stream of water is kept running through, or armed sentinels are placed on guard.

Seam diggings.—In some districts, innumerable small stringers and seams of quartz, frequently much decomposed, and all carrying more or less gold, are found permeating a soft easily-disintegrated slate formation ; and it has been proved feasible to treat this formation by hydraulic washing, as the gold has been thoroughly liberated from its matrix by natural processes. Hydraulic is conducted on somewhat different principles in the case of seam diggings. The sluices are set at a steep grade, even 18 in. per 12 ft. ; the whole formation cannot always be washed away indiscriminately, for sometimes barren earth only would be passing through the sluices, and at other times the gold might be too plentiful to be properly caught. The pay-dirt does not run horizontally, as in gravel deposits, but vertically, in a narrow channel of quartz seams related to a well-defined wall or fissure, which always pitches at a steep angle, and often dissects all the veins on one side. The auriferous rock is not always closely confined between two perfect walls, but is often spread out 20 to 50 ft. on one side of the main fissure. It is commonly associated with a series of lenticular masses of quartz, crossing or lying parallel with each other, and having the same dip ; also in the form of chimneys, where courses of seamlets cross each other. There is a great tendency to form lenticular masses, often measuring only a few feet each

way, but occasionally 40 ft. long and wide, and a few feet thick. When near the surface and the ground is decomposed, and the pay-seams are numerous and widely distributed, it is best to hydraulic away the whole hill ; but as soon as the deposit is beyond the reach of water, the seams must be followed by shafts and drives, and the working generally assumes the character of an ordinary vein-mine, the stringers becoming harder and more strongly marked as the region is reached where surface influences have not been at work.

A somewhat similar plan was adopted by Stackpoole in dealing with a soft granite dyke containing veins of poorly auriferous quartz, at Wood's Point, Victoria, for an account of which the author is indebted to A. B. Ainsworth, C.E., of Alexandra. Water to the extent of 6 or 8 sluice-heads (say 1200 to 1600 gal. a minute) was available at an altitude of about 80 ft. above the creek draining the outcrop of the dyke. The loose stuff was run through sluice-boxes with false bottoms, and the quartzose débris was crushed. The lumps of quartz were turned into a paddock at the end of the sluice-boxes during the sluicing operation, by means of a "grizzly" made of timber rails 3 or 4 in. apart. A couple of forkers were employed in the paddock pitching away the stones and rocks known to be valueless. In the course of 2 or 3 weeks, some 300 or 400 tons of material accumulated in the paddock ; this, trammed to the stamps, and crushed at an average cost of 1s. 6d. per ton by water-power, yielded 2 dwt. of gold per ton, in addition to the loose gold caught in the sluice. About 9 men were occupied in the sluicing and paddocking.

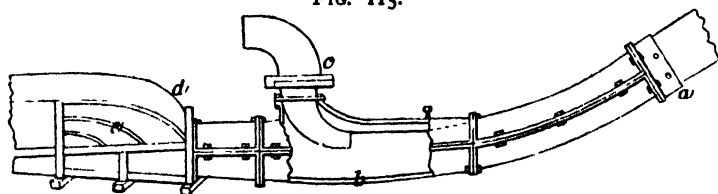
Crushing process.—In some places where water has not been found in sufficiently constant abundance, and also where difficulty has been encountered in finding outlet for the tailings, hydraulicing has been superseded by other methods. One of these consists in removing not the whole depth of the auriferous ground, as is necessitated by hydraulic operations, but only a few feet of the richest stratum from the bed-rock upwards. This permits a portion of the gold to be extracted from ground which must otherwise lie unworked, and in some instances the immediate return for outlay is much greater than would be the case with hydraulicing ; but the less auriferous ground left will probably be found too poor to pay for working alone by hydraulic or any other process at a future date. The pay-dirt extracted has to be put through a crushing process in order to liberate the gold. Raymond mentions a Californian company as the only one working successfully in this way, and gives the following particulars of their operations :—men employed, 25 ; wages, 12s. per day ; output, 40 to 50 carloads per 24 hours, according to nature of ground ; motive power, water—75 in. with a head of 80 ft., projected against a 10-ft. hurdy-gurdy wheel.

Average yield per carload (19½ cub. ft.),	<i>s.</i> 15	<i>d.</i> 8½	cost,	<i>s.</i> 8	<i>d.</i> 5½	profit,	<i>s.</i> 7	<i>d.</i> 3
„ „ ton of 2000 lb.	„ 19	7½	„	10	7	„	9	0½
„ „ cub. yd.	„ 22	0½	„	12	1	„	9	11½

Drake's cement-mill (see p. 226) is sometimes applied to this purpose. It is so set in connection with the sluice-boxes that such portion of the gravel as may have become fine by the washing and blasting flows through a grizzly leading under the mill to the opposite end, while the hard gravel passes into the mill ; it is there disintegrated, discharged into the sluice, and passed to the under-currents. This obviates the necessity of having "drops" to break up the material. It is stated that "the whole expense of working cement by this machine, when put in with cars, is about 25c. (12½*d.*) per sq. yd., when working 1000 to 2000 tons per diem. When put in through the sluices, some 5000 tons or more a day can be put through at a cost of between 2c. and 3c. (say 1½*d.*) per yd."

Hydraulic elevators.—Cranston's hydraulic elevator (Fig. 115) is devised to overcome the difficulty of washing gravel lying below the natural outlet of the basin that contains it. A closed flume *a* is placed on an upward incline, and fitted with a horizontal one *b* of cast-iron, into which is introduced a hydraulic nozzle *c*, pointing in the direction which the gravel is required to take. The horizontal portion of the apparatus has an open end *d* behind the nozzle, by which the gravel and water enter. The gravel is washed in as it might be into any other flume, but a grizzly *e* is arranged to turn off the large stones at the mouth. The gravel and water pass from the closed flume *a* into an ordinary sluice at the top, and

FIG. 115.



CRANSTON'S HYDRAULIC ELEVATOR.

it is said that they have been run through the flume as thick as the water will carry the gravel through a sluice set on a 24-in. grade, and that it was found impossible to choke the apparatus by any head of water it was calculated to carry ; also that it washes the gravel as well as ½ mile of ordinary sluice would do. Under-currents are placed at a short distance down the sluice from where the elevator-spout discharges into it, drawing all the heavy materials—gold, mercury, &c.—from the sluice as soon as it has time to settle from the agitation produced by the discharge. This under-current is so near the head of the sluice that it saves most of the

gold, and can be cleaned up every day if desired, without stopping the apparatus. To recover the elevating water, and re-use it for driving the gravel into the machine, a settling-tank is made on the side of the sluice, just far enough down from where the spout discharges to allow the surface water to become settled a little. The side of the flume next the settling-tank is cut away, and bars are placed lengthwise at short intervals apart across the opening into the flume. The surface of the water containing but little gravel flows over into the settling-tank. The bars prevent the stones and boulders from passing over, and keep them in the main flume. It is claimed for the process that wherever hydraulic pressure can be had—say 100 to 300 ft.—a mine can be worked 20 to 50 ft. deep to as good advantage as through an open cut or tunnel.

The following general instructions may be useful. The machine should be set 5 ft. deep in the bed-rock, in a timbered room or box. The water-lifter should be set on one side of the box, 2 ft. lower than the elevator, and should be set before the elevator, and used as a pump and to run out the dirt, gravel, &c., in the excavation where the elevator is set. The elevator should be set on bed-pieces put crossways of the shaft or box. The box should be for a 12- or 16-in. machine, 12 ft. long and 6 ft. wide on the clear inside, so as to allow of turning a wrench between the machine and the side of the box. For a 20-in. machine, it should be 16 by 7 ft. inside. The water-lifter should be set outside of this box. The incline of the discharge spout is 45° ; 17 in. in length of this spout make 1 ft. high. This spout should be made of 3-in. plank, and large enough for a set of blocks to go inside; and the top or lid should be held on by clamps, and not nailed; the bottom and two sides should be nailed together with 6-in spikes. The spout should have a curve at the top, so as to discharge straight into the flume. This curve should be 8 ft. long, and should rise in its length 3 ft. for a 12- or 16-in. machine; for a 20-in. machine, it should rise 4 ft. in its length. In calculating the length of the discharge spout, deduct this amount; also deduct the curve in the discharge end of the machine, which is as follows: for a 12-in. machine, there is a rise in the machine itself of 18 in.; in a 16-in. machine, there is 26 in. rise; in a 20-in. machine, there is 4 ft. rise in the machine—so this much, with the curve on the machine, should be deducted from the amount required to rise, and the spout be made accordingly. The grizzly or feed-box should be—for a 12-in. machine, 3 ft. 9 in. long; for a 16-in. machine, it should be $4\frac{1}{2}$ ft. long; and for a 20-in. machine, it should be $5\frac{1}{2}$ ft. long. Its grade, should be, in its length—for a 12-in. machine, 10 in.; for a 16-in. machine, 12 in.; and for a 20-in. machine, 16 in. The flume should be—for a 12-in. machine, 30 in.; for the top flume, with whatever grade the ground will admit of, giving sufficient dump. The feed-sluice in the bottom should be 24 in. wide, and should



IN-PHOTO, BRUNNEN & OF LONDON.

BLUE SPUR. BRITISH AMERICAN CLAIM.

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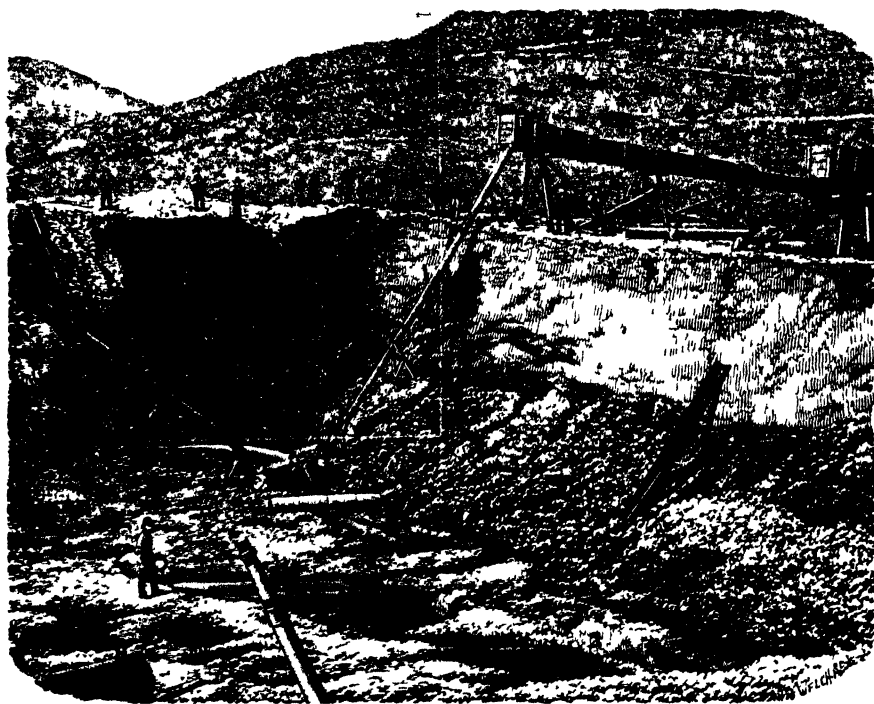
have at least 6 in. grade to 12 ft., and as much more as the ground will admit of. These grades will do for all the larger sizes; but the size of the flume should be as follows: a 16-in. machine wants a 3-ft. discharge and a 30-in. feed-flume; and a 20-in. machine wants a 4-ft. discharge-flume, and a 3-ft. feed-flume in the bottom. The sizes of "giants" required are as follows: a 12-in. machine wants a No. 2 giant; a 16-in. machine wants two No. 2, or one No. 3 or 4 giant; and a 20-in. machine wants two No. 4 or 5 giants. The main pipes should be the size of the machine or larger—that is, a 12-in. machine should have a 12- or 15-in. main; a 16-in. machine should have a 16- or 18-in. main—18 in. would be better; a 20-in. machine should have a 20- or 22-in. main. Forks or crotchets are better than distributors for the lower end of the main pipes, and want regular screw water-gates.

Concerning Cranston's elevator, Prof. Skidmore obligingly writes that it is most successfully used in working ground with a light grade. There are at least a dozen in use in California alone. He has not seen any treating gravel "considerably below the ground level." Those seen were dealing with ground about 3 ft. in depth, where the grade was so light that water would scarcely run. The elevator raised the gravel in one instance 18 ft., and in another 13 ft. to the sluice-box. Wm. T. Dewey says that a water-pressure of 180 ft. (miners' measurement) will give a lift of 15 ft., and 300 ft. a lift of 25 ft., and so on. Skidmore states that where he has seen the elevator in operation, the amount of water used has been 160 to 240 "inches" and the lift 13 to 23 ft.

Perry's elevator.—The working of Perry's elevator is thus described by Warden Carew. The works consist of a drainage-race pipes, to lead head-water, an elevating-pipe, and sluice-boxes. The drainage-race is made of sheet iron, forming a pipe 30 in. diameter. At the lower end of the race, this pipe is laid on the surface of the tailings, and it is extended up the gully through a channel opened in the tailings, with just sufficient inclination to allow water to drain down the gully. This pipe is continued for a distance of 1500 ft. up the claim, and is at that point a considerable depth from the surface of the tailings and 2 or 3 ft. into the bed-rock. By means of this race, the object sought for—thorough drainage—is effectually secured, and the race can now be extended as the workings advance up the gully, by simply cutting a channel in the rock, the distance and depth required, and covering it with suitable material. Near the present head of the drainage-race, operations at sluicing the ground have been commenced. A large open space was first made down to the rock bottom, and the elevating apparatus placed in position. This consists of a double pipe 15 in. diameter, made of sheet iron, joined with a half circular cast-iron elbow of larger diameter than the pipe, and perforated

at the angle with an aperture of 12 or 14 in. diameter. The elbow is placed in a hollow, or shallow well-hole, in the rock, and the ends rise nearly perpendicular 12 or 15 ft. above the surface of the tailings. One end of this pipe is joined to another pipe of the same diameter, which conveys water from a supply-dam situate at an elevation of 294 ft., thus affording great pressure. The other end of the pipe is made to discharge into the head of a long line of sluice-boxes supported on trestles over the bed of the gully. When water is let in from the supply dam, it passes down through the water-pipe, and then rushes up the elevating-pipe,

FIG. 116.



PERRY'S HYDRAULIC ELEVATOR AT WORK.

drawing, as it does so, through the aperture in the elbow, all gravel, cement, earth, or water within reach. With a sufficient supply of water, the process seems capable of lifting an enormous quantity of stuff, and it is projected up the pipe with immense velocity, estimated at a mile a minute, by which means all unbroken lumps of cement strike a thick plate of iron placed for that purpose at the top of the pipe, and, by force of the blow, become pulverised. Everything reaching the head of the pipe then flows through an aperture into the sluice-boxes, which, being raised as explained, can be so arranged as to discharge *débris* at any

locality or distance, as occasion may require. Another appliance is a line of 15-in. piping, about 3000 ft. in length, which conveys water from a lagoon at the head of the gully to near the face of the workings. The pipe is there fitted with a patent nozzle, with which the water is played upon the face, sides, and bottom of the workings, and forces the earth and tailings in large quantities towards the aperture in the elbow of the elevating-pipe, whence all is drawn up the pipe and deposited in the sluice-boxes. When the claim is properly opened out, and the process complete, the gravel and earth will be first elevated on to a platform by means of buckets travelling on a band, worked by a turbine wheel, and be there discharged on to a grating for the purpose of separating all large stones; these will fall on to a travelling chain-belt, and be deposited in any convenient spot out of the way of the workings. The fine stuff will fall through the grating on to a hopper at the base of the elevating-pipe, be drawn through the aperture, and lifted a further 20 to 40 ft. as may be necessary, to afford sufficient space for deposit of débris from the sluice-boxes. The apparatus is shown in full work in Fig. 116.

Booming.—The term “booming” is applied, in the Western States of America, to an operation much resembling the old process of “hushing,” as used in Yorkshire for discovering lead-lodes. A reservoir is first constructed at the head of the ground to be worked; into this water is conducted, from the most convenient source still higher up, by flumes or ditches. These reservoirs vary in size from a small pond to an acre or two lake, and the ditches are often 8, 10, and 12 miles long. When the basin is full, and a continuous head of water is in running operation, gates are opened, letting loose the whole volume of the liquid, which tears down the mountain side in a huge volume, sweeping everything before it, carrying tons of boulders, gravel, and dirt down to the gulch below. If auriferous ground is to be worked, a long and massive wooden flume is built at the foot of the hill, into which the débris is carried, with all the force of the falling waters and the sand and rocks washed along in its course, while the gold is deposited by its own gravity, behind the riffles in the bottom of the race. These flumes are often thousands of feet long, and as rocks of all sizes and weights are carried along in them, they must be built with great strength and solidity, to withstand the immense wear.

The self-acting gate, now considered the best (whereby the opening and shutting of the gate of the reservoir is made automatic), consists of a water-box suspended in guides, the rope from which passes over two pulleys, one of 12 ft. and one of 5 ft., to the lower edge of the canvas gate (barred with strips of iron or 2-in. timber). When the water in the reservoir reaches the proper height, a small flume conducts it to the box, which, when full of water, has weight enough to roll up the canvas gate

at the bottom of the reservoir from the bottom, allowing the water in the reservoir to issue through a gate (generally 4 x 6 ft. in size). By the time the reservoir is nearly empty, the water in the weight-box has discharged itself through holes made for that purpose in the bottom, and a weighted arm on the second pulley drops the gate to its place, when the pressure of the water keeps it there water-tight. One man is now considered ample force to run a boom, and his duties consist mostly in clearing timber from the ground to be worked and in breaking the larger boulders into sizes small enough to go through the flume, which is usually 4 ft. wide with a grade of 1 ft. in 12 ft. The use of a boom permits the working of ground that could by no other means be made to pay. The experience of the Summit County miners goes to prove that, notwithstanding the large amount of water used and the velocity with which it rushes through the flume, the gold collects readily in the upper boxes of the flume, in which mercury is generally placed.

De Groot observes that the plan of filling reservoirs and then hoisting the gates and letting the water escape in a flood, as practised by the ancients, has its counterpart in the "self-shooter," so called, a contrivance through the use of which the opening and shutting of the reservoir gate becomes automatic. These devices are in use in California, and more extensively in certain parts of Colorado, where this method of collecting and discharging water is called "booming." This self-acting gate, which is not much employed, except in localities where water is scarce and labour requires to be closely economised, may be described as follows : To the upper edge of the gate, which consists of a heavy piece of canvas, a rope is attached that passes up over a 5-foot pulley fixed above the dam, and thence back 15 feet over a much larger pulley, dropping down from which its end is fastened to a large wooden box or cistern. A small flume extends from the upper edge of the reservoir to this box ; when the reservoir is full, the water flows out through the flume into the box, which, as it fills up, moves downward, raising the canvas gate and letting the water out into a race which carries it down to the mine. Meantime, the water in the box having escaped through holes in the bottom made for the purpose, the box ascends, being raised by a weighted lever attached to the axle of the smaller pulley. With this action, the canvas gate drops again to its place, and the flow of water being here arrested, the reservoir fills up, once more repeating the process above described. The advantages of this mode of "booming" are, that all the water is saved, while the services of a gate tender are dispensed with. The above is the style of this device as rigged up in Colorado, a long sweep and wooden gate being, in California, substituted for the canvas gate, rope and pulleys.

Utilising River-currents for Sluicing.—J. T. Thomson suggests the

following method of utilising the current of large rivers for gold-slucing. A screw 8 ft. in diameter will give 8 H.P., and can be immersed and attached to any fixture. It will raise one Otago sluice head of water (equal to .95 cub. ft. of water delivered per minute) to an elevation of 70 ft., or 7 heads 10 ft., without intermission. The screw may be made of timber, and can be put together by any blacksmith or carpenter. Minor scientific faults in its form are compensated by an excess of current-power. The accessories requiring skill in their construction are a brass force-pump and some india-rubber tubing. With a screw or fan 15 in. in diameter, with blades set at an angle of 20° to the disc, in a 2-mile current, the revolutions are nearly 1 per second; the pistons of the pump are worked by a crank, propelling the contents once per second; the diameter of the cylinder is $\frac{3}{4}$ in.; stroke of piston, 2.7 in.; quantity of water per stroke, 1.1925 cub. in., or 71.55 cub. in. per minute, or 59.5 cub. ft. or 368.9 gal. per 24 hours; cost 4*l.* or 5*l.* The capacity and cost increase, of course, with the size. The whole apparatus is easily repaired, and can be removed at will.

Drawbacks to Hydraulicing.—The main difficulty with hydraulicing is to find an outlet for the refuse, and were it not for this, there would be greater activity in this description of mining. The difficulty, fortunately does not affect all mines, as some have facilities for disposing of their "tailings" in valleys or gulches, without making use of rivers or creeks. One Californian company has spent a large sum of money in purchasing the land they required for their débris, and others no doubt will do the same. Two or three suits have been brought against hydraulic mining companies by farmers, who have had their land deteriorated in value, or rendered entirely valueless by the deposit from the mines. Another serious question cropping up is as to the damage done to the navigation on the rivers and bays. This question has received the attention of the Federal authorities in Washington, at the instance of the Chief of Bureau of Engineering, who has reported that the arm of San Pablo Bay, separating Mare Island from Vallejo, and forming the Navy Yard Harbour, is filling, and that he attributed it to the system of hydraulicing.

The result of the legal war which has been going on for some years between the hydraulic miners and other interested parties in California is, that the government has stopped hydraulicing in all except the Klamath river and its tributaries.

Working Results.—Under favourable conditions for hydraulic washing two men can do all the work required, in a washing that uses 300 miners' "inches" of water. Under such circumstances, 1 pipe will break down as much as 3 can wash away; on the other hand, 3 pipes are sometimes required to break down what 1 can wash away. The water is generally considered capable of carrying away $\frac{1}{3}$ of its own weight of gravel.

The average cost for the La Grange Company, according to Bowie, may be roundly stated as follows :—

Per oz. of Metal produced.			Per cub. yd. treated.		
	<i>s.</i>	<i>d.</i>			<i>d.</i>
Water	5	10	Water		0·4
Labour	28	4	Labour		1·8
Material	7	6	Material		0·5
Officers	3	11	Officers and contingent	}	0·3
Contingent expenses ..	1	1	expenses		
Taxes	0	4			
	47	0			
					3 0

The above refer to light pressure and low grade. For heavy pressure and 4 per cent. grade, an example may be found in the North Bloomfield Company's figures, per oz. of metal produced :—

	<i>s.</i>	<i>d.</i>
Labour	16	0
Blocks and lumber .. .	2	0
Explosives	4	0
Materials	3	8
General expenses	2	11
Water	8	0
	30	7

The following promiscuous costs per yd. may also be quoted :— $3d.$; $1\frac{1}{2}d.$; $1\frac{1}{4}d.$; $1d.$; $1d.$; averaging from $1d.$ to $2\frac{1}{4}d.$ per cub. yd.

Of the yield from hydraulic workings, those lying between the Middle and South Yuba have averaged, according to Laur, $8d.$ per cub. yd. and according to Prof. Silliman, $15d.$ per cub. yd. Various yields have been :— $2s.$ $11\frac{1}{2}d.$; $2s.$ $6d.$; $12\frac{1}{2}d.$; $7\frac{1}{2}d.$; $6\frac{1}{2}d.$; $5d.$; $2\frac{1}{8}d.$ The last-mentioned figure refers to the Gold Run district, where all the conditions were extraordinarily unfavourable. The average yield of the Smartsville gravel is stated by Whitney at about $23c.$ ($11\frac{1}{2}d.$) per cub. yd.

The hydraulic method has been in use on the Sierra Nevada over 20 years, and the experience of this period affords a means of judging the value of the gravel and the profit in working it. The general results have been very satisfactory. Wherever the richer blue gravel has been accessible, as at the Flats, Badger Hill, and below San Juan, it has, with very rare exceptions, paid profits, and sometimes large profits, to its owners. The top gravel, though much poorer than the blue, has often been found very rich in streaks (due to concentration by surface streams) and has, in general, paid large sums of money to the ditch companies furnishing the water, leaving something besides for the owners of the ground. There are few trustworthy records showing in detail the costs and profits or losses of the business in the earlier years of hydraulicing ; but so far as the top gravel is considered, the price paid for the water used in mining is some indication of the result obtained.

In early years, the price of water was 25c. ($12\frac{1}{2}d.$) per "inch" for 10 hours' flow. This price has fallen, by gradual reductions, to 20c., $16\frac{2}{3}c.$, $12\frac{1}{2}c.$, $10\frac{1}{2}c.$ per "inch" for 10 hours, the price at present varying from 8c. ($4d.$) to $12\frac{1}{2}c.$ ($6\frac{1}{4}d.$) per "inch" for 10 hours, or twice that price for 24 hours. In many claims, in which top gravel only was washed, the water was paid for at 20c. to 25c. per "inch" for 10 hours, and instances are reported in which, after paying these charges, the owners retained handsome profits; such cases were, however, exceptional. On the other hand, it is well known that under the high rates charged for water in early days, many attempts to wash the top gravel resulted in loss.

At the present day, where the top gravel only is washed, it is thought to do very well if it yields 10c. to 15c. ($5d.$ to $7\frac{1}{2}d.$) per "inch" of water for 10 hours. On the North Bloomfield Co.'s mine in 1870-71, the yield of surface gravel was 16c. ($8d.$) per "24-hour inch," or $6\frac{1}{10}c.$ per "10-hour inch." From 1870 to 1874, the yield was only $13\frac{1}{10}c.$ per "24-hour inch," equal to $5\frac{1}{6}c.$ per "10-hour inch." In 1875, the top gravel, including a little blue gravel, but nothing within 40 ft. of the bed-rock, yielded $19\frac{2}{10}c.$ per "24-hour inch," or 8c. per "10-hour inch." It is quite probable, however, that the water in this mine, furnished as it is from the company's own ditch, is much more lavishly used than in mines where it is purchased, and the relation of water to product would, on that account, be an unfair criterion for other mines. At Columbia Hill, where only top gravel has been washed, its yield in several instances has varied from about 20c. ($10d.$) to 58c. ($29d.$) per "24-hour inch," affording in the instance last referred to exceptionally good profits.

Some of the mines below San Juan afford the most satisfactory examples of the results of washing the entire bank, including together the top and bottom gravels. The record of their observations during four years, where they can be obtained, furnish the best data for judging the value of the ground. Of this portion of the channel, some 2 or 3 miles have been already washed out. The American mine has been worked for a length of about 3000 ft. along the channel. The width of the cut, from rim to rim, is probably 1000 ft., and the workings about 140 or 150 ft. From some data furnished from the company's books a few years since, it appears that the gross product of the mine from December 9, 1860, to August 6, 1872, was \$1,241,240 30c. ($258,591l.$ $14s.$ $7d.$). The water used in this period was 1,454,174 "inches" of 10 hours, and the yield per "10-hour inch" would accordingly be 80c. ($3s.$ $7d.$). The price paid for the water varied from $16\frac{2}{3}c.$ to $12\frac{1}{2}c.$ per "10-hour inch," amounting in the aggregate to \$218,749 58c. ($45,572l.$ $16s.$ $7d.$), or 30·6 per cent. of the whole working expense. The last-named item was \$714,771 4c. ($148,910l.$ $12s.$ $8d.$), and the nett proceeds \$526,469 27c. ($109,681l.$ $1s.$ $11d.$), or 42·41 per cent. of the product. In 1871, Hamilton Smith estimated that the yield of this ground amounted to

24c. (1s.) per cub. yd., and that each linear ft. of channel worked had paid at least \$750 (150*l.*).

These examples might be supplemented by others, but enough has been said to warrant the conclusion that the top gravel alone usually contains gold enough to pay the expenses of mining, and leaves a profit for the owner of the water ; and, further, that where the blue or bottom gravel has been washed, it has, with very rare exceptions, made satisfactory profits for the owner ; finally, in order to reach the blue gravel the top gravel must be removed.

Four companies at Howland Flat and Potosi—the Down East, Union, Hawkeye, and Pittsburg—took from about 2,365,000 sq. ft. of surface, \$2,251.65 395c. (470,094*l.* 6s. 5*d.*), the pay-gravel being estimated at 4½ ft. in thickness. This would give an average of 95c. (3s. 11½*d.*) per sq. ft. of surface, or \$5 70c. (23s. 9*d.*) per cub. yd. of gravel washed. This material was mined, it is stated, at a cost of 47c. (23½*d.*), leaving a profit of 48c. (2s.) per sq. ft. At Grass Flat, in the Pioneer Company's ground, the yield per cub. yd. of gravel is said to have been \$1 59c. (6s. 7½*d.*).

At Allan's Flat, Yackandandah, Victoria, the ground washed off was about 30 ft. deep, the water used was 500 gal. a minute, and the inclination of the sluice was 1 in 25 ; 3 men worked 150 cub. yd. per diem. The daily expenses of a claim were :—

	£	s.	d.
3 men at 8s.	1	4	0
360,000 gal. of water at 0.33 <i>d.</i>	0	10	0
Wear and tear	0	6	0
	<u>2</u>	<u>0</u>	<u>0</u>

Consequently a yield of ½ gr. of gold per cub. yd. covered all expenses.

A hydraulic claim on Dunedin Flat, Kumara, New Zealand, yielded 1735 oz. of gold from a block of 22,403 cub. yd., or an average of 31 gr. per cub. yd., working a face 35 ft. deep. Shares in this claim sell readily at 400*l.* each.

The cost of producing 1 oz. Troy of gold in the Californian hydraulic claims is given by Egleston as follows :—

	La Grange Co.			North Bloomfield Co.		
	\$	£	s. d.	\$	£	s. d.
Water	1.43	0	5 11½	2.09	0	8 8½
Labour	6.85	1	8 6½	3.93	0	16 4½
Materials	1.81	0	7 6½	0.88	0	3 8
Explosives	..	..	..	0.98	0	4 1
Blocks and lumber	..	..	..	0.50	0	2 1
General expenses	0.94	0	3 11	0.70	0	2 11
Contingent expenses	0.26	0	1 1	..	..	..
Taxes	0.09	0	0 4½	..	..	..
	<u>\$11.38</u>	<u>£2</u>	<u>7 5</u>	<u>\$9.08</u>	<u>£1</u>	<u>17 10</u>

The value of the metal is \$18.53 (3*l.* 17*s.* 2½*d.*) per oz.

The height of bank washed down, and the yield of several mines, is stated by the same author, thus:—

	Height of bank	Yield per cub. yd.	
	ft.	cents.	<i>d.</i>
Smartsville Claims, Yuba Co.	112	19.5	9½
Blue Tent, Nevada Co.	180	15	7½
North Bloomfield, Nevada Co.	180 to 260	4 to 6.5	2 to 3½
Gold Run, Placer Co.	200	4.8	2½
Columbia Hill, Milton Co.	100	4.33	2½
La Grange, Stanislaus Co.	18 to 100	2.5 to 15.5	1½ to 7½
Patrickville, Stanislaus Co.	40 to 60	4.33 to 18.5	2½ to 9½
Dardanelles, Placer Co.	150	13	6½

The cost and yield per cub. yd. of some of the mines is :—

	Cost.		Yield.	
	cents.	<i>d.</i>	cents.	<i>s.</i> <i>d.</i>
Roach Hill.. . . .	6	3	60	2 6
Richardson.. . . .	3	1½	15	0 7½
Iowa Hill	2.5	1½	71	2 11½
Independence	2	1	25	1 0½
Wisconsin	2	1	12.5	0 6½

The drift of the Nine Mile Creek diggings, New South Wales, taken as a whole, is of a very favourable character for hydraulic sluicing. It had none of the great difficulties to be met with in some deep leads, where occur two thick seams of basalt, which, when undermined, tumble down in large blocks, occasioning a great loss of time to remove out of the way; it disintegrated easily, and, with the exception of the lignite and a few of the heaviest of the boulders, was easily disposed of by the existing limited method of sluicing. With a water-supply of 6,000,000 gal. daily, under a pressure of 140 ft. or 60½ lb. per sq. in., Gipps estimated that at least 1500 cub. yd. could be sluiced away per diem. Taking the yield per cub. yd. at only 10 gr., value 1*s.* 6*d.*, the yield per diem would be 112*l.* 10*s.*, amounting to 28,350*l.* per annum of 252 days, which would allow 10 weeks for washing up the tail-race, and for stoppages. The working expenses should not exceed 2350*l.* The area sluiced away every year by this estimate would amount to about 2½ acres.

Losses.—The basis on which hydraulic mining is pursued, at least in America, is that it pays better to treat a great quantity of gravel with a great loss of gold than a small quantity with a small loss. The consequence is that the waste of gold is really enormous. The coarse gold is probably caught pretty effectually; but the greater part of the fine and rusty gold is undoubtedly lost, and it is not too much to say that 15 to 20 per cent. of the total amount of gold present in the gravel is by this system scattered throughout the watercourses of the country in such a way

that it is lost, not only to the miner of to-day, but to succeeding generations. One party who took the trouble to wash some of their tailings in a rocker got \$5 (1*l*.) worth of gold from 60 bucketfuls of tailings, or an average of 4*d*. per bucket. This loss does not consist either of gold alone. Of the mercury put into the sluices for amalgamating with the gold, 10 to 25 per cent. passes away with the tailings. This is due partly to the flouring of the mercury, from the constant attrition to which it is subjected; but is also owing in a great measure to carelessness in watching the sluices and keeping them in repair. Instances are not unknown where many pounds of mercury have been taken from a single pocket where it had gathered in the tailings. Considerable profit has been derived in some cases from rewashing deposits of tailings, the conduct of the process being rendered vastly cheaper than other sluicing by reason of the abundance of mercury present.

As to the loss of gold in the operations carried on in the Smartsville district, in the opinion of some well-informed miners, it is not over 20 per cent., while others set the amount as high as 50 per cent., and express a belief that the bars of the Yuba, into which the tailings are run, would be found to be as well worth working as they were in 1849. One most experienced and reliable authority declares that not more than 33 per cent. of all the gold contained in the gravels is saved. During very cold weather, much gold which comes into actual contact with the mercury escapes again, owing to the small degree of affinity at low temperatures.

The State Mineralogist of California, in a recent report, gives expression to the following views on the question of losses in hydraulic.

"It is my opinion that the methods now employed in hydraulic mining should be materially modified. As far as the disintegration of the gravels is concerned, nothing could be better, but the very force used in this operation defeats the object if carried too far. I consider it a foregone conclusion that coated gold cannot be saved by the use of quick-silver alone, and that it, with much of the fine gold, is carried forward by the great force of the water in the sluices and lost in the dump. This force, necessary to wash down the gravel banks, is not desirable in the sluices; on the contrary, it is highly detrimental, if the great mass of the worthless material can be got rid of at an early stage of the operation, as practised on a small scale in pan washing. It is only to carry this débris through the sluices that this powerful current of water is required.

"It is easy to find fault with the working of any process in use, but not so easy to propose a remedy. What I am about to suggest is purely theoretical, and may not work in practice, but it is worthy of the consideration of miners. The 'grizzly' is used in nearly all hydraulic mines to sift out—on a large scale—the boulders present in the flowing débris, and it is a recognised principle to do this as early as possible. The débris freed from the larger boulders passes through the

sluices more freely, after which it is caused to flow over the pavements and dropped on the bedrock. But there is a point beyond which this treatment should cease. The proper point being decided upon, a set of grizzlies should be introduced, diminishing in fineness until the last one is no wider between the bars than twice or thrice the width of the largest particles of gold found in the mine, as learned by experience. What remains on the grizzlies should be conveyed to the dump by a special channel, using a large portion of the available water for that purpose, or, when possible, shot off into some lateral cañon or depression. To divide the water, and, at the same time, to allow a portion to pass through with the fine gravel and sands, the bars of the last grizzly should be laid at right angles with the current and given the proper grade, which can only be learned by experience. What passes the last grizzly should be conveyed—using the remainder of the water—through sluices, as at present, but with less grade, and so constructed that a considerable portion of the heavier matters should remain in them, including the gold, arrested by properly constructed riffles. From this point it might be necessary to have double sluices, one to be used while the other is being cleaned up.

“At proper periods, as determined by experience, the accumulation should be removed and concentrated, using either of the well known concentrators for that purpose. Rusty gold would thus be saved by its specific gravity, which is known to escape the quicksilver, while fine gold, moving more slowly in the less inclined sluice beds, would find its way to the bottom and be saved. This is attempted now in the undercurrents, but the fault with them is, that there is too little space provided for the accumulation of the heavier parts of the material washed, and the absence of the secondary concentration. Small boulders are now rushed through the whole length of the sluices, and dumped into the rivers, miles away. It is absurd to suppose that gold should not be carried forward with this rush of water, and especially so after the interstices between the pavement blocks are filled.

“At the American mine, Nevada County, there are twenty undercurrents, and gold is saved on the last one. Is it supposable that another would not also save a portion of the precious metal? It is a well known fact that diamonds exist in the gravels of some of the deep placers, and it may be assumed that they are also in others not now known, yet it seldom occurs that one is found, which I consider another proof that too great a force of water is employed in the sluices below the bedrock tunnels.

“This proposed improvement is clearly based on the results of my experiments, and it seems to be theoretically correct, but how it will meet the views of the experienced miners, or how it will work in practice, remains to be proved.”

CHAPTER VI.

MINING GOLD-BEARING VEINSTUFF, AND EXTRACTING THE FREE
GOLD FROM THE STONE.

THE occurrence of gold in mineral veins and lodes has already been discussed from its geological and mineralogical standpoints (see Chapter I.). It remains to consider here some peculiarities in the veins, which have a bearing upon the mode of mining, as well as the whole subject of the nature of the association between the gold and its matrix, and the means adapted for extracting it from that matrix.

Sections of Veins.—There would appear to be no actual limit to the variety of forms exhibited by auriferous veins of quartz and other minerals, and it would require a large volume to describe even a portion of the variations met with in any one mining district. It must suffice to illustrate some few of the most remarkable examples, which may be done within a convenient space.

FIG. 117.

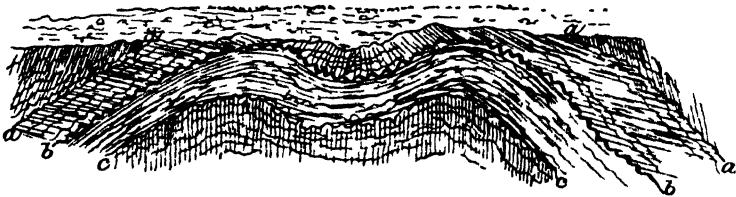


FIG 118

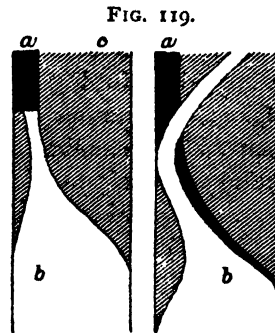


CONTORTED VEINS IN THE CARIBOU DISTRICT.

Fig. 117 shows a contorted vein worked in the Caribou district of Nova Scotia. It consists of an anticlinal fold having a subordinate synclinal fold at the apex. The dip of the axis of the anticlinal is east. The gold is followed by stopes, and the quartz is removed by a process

of long wall. The yield is about 4 oz. of gold per ton. The veins *a c* are about $\frac{1}{8}$ to $\frac{3}{4}$ in. thick; *b* varies from $\frac{1}{2}$ in. to 4 in. The country-rock is a soft slate, in places a sandy shale; the slate underlying *b* is auriferous in patches. Fig. 118 illustrates more clearly the curious contortions of the vein *b*.

Daintree says of the reefs of the Cape river district, Queensland, that they are situated on a flat at the foot of a high ridge, which follows the course of an elvan dyke of quartziferous porphyry. Branch veins from this intrusive mass have been injected into the surrounding slates, and it would almost seem that, in some cases, the rich quartz veins were a continuation to the surface of the elvan veins themselves. Sections (Fig. 119) taken in one of the shafts in the reef appear to warrant this assertion: *a*, reef; *b*, elvan dyke; *c*, slates.



ELVAN DYKES.

FIG. 120.

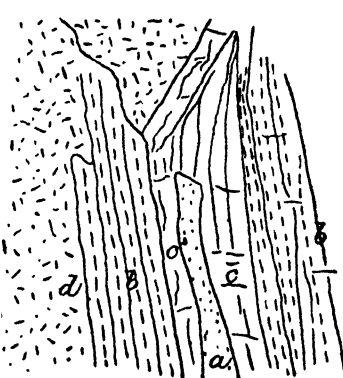


FIG. 124.

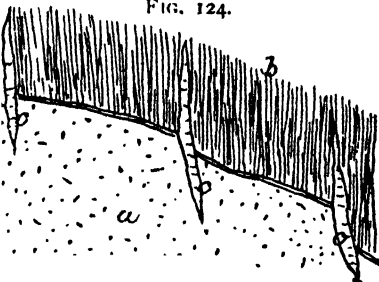


FIG. 122.

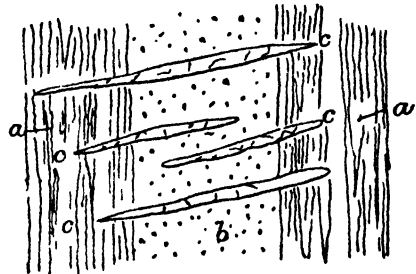


FIG. 123.

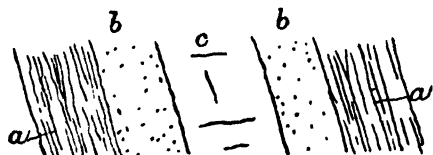
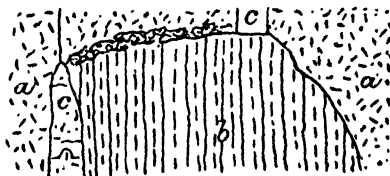


FIG. 121.



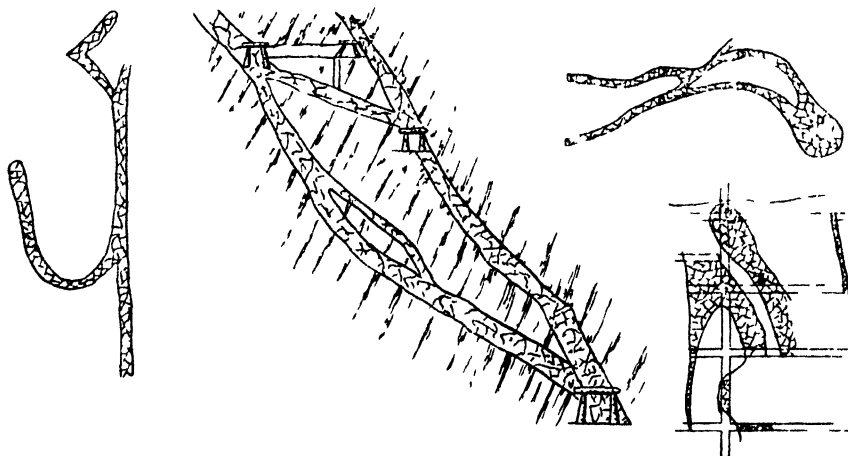
QUARTZ VEINS IN DIORITES AND GRANITES.

Figs. 120 to 124 illustrate the occurrences of quartz veins in the diorites and granites, and their contact zones, at Swift's Creek, Victoria.

In Fig. 120, *a* is a horse of silicious felsitic rock, containing much ordinary and arsenical pyrites; *b*, finely crystalline foliated contact-rock, with a little pyrites, and very micaceous in places; *c*, white translucent vein-quartz, about 8 in. wide, with arsenical pyrites, yielding about 1 oz. per ton; *d*, quartz-diorite. In Fig. 121, *a* is coarse gneissic quartz-diorite, very much decomposed; *b*, schistose hornfels; *c*, auriferous quartz vein, 4 in. wide. In Fig. 122, *a* is Silurian (? Upper) mudstones; *b*, granitic dyke; *c*, auriferous quartz veins. The veins pass through the dyke and penetrate the sedimentary rocks on each side; the dyke varies in width from a few feet to 200 ft., and is almost completely decomposed. In Fig. 123, *a* is Silurian (? Upper) slates; *b*, dyke; *c*, auriferous quartz veins. The dyke is a compact or fine-grained diorite of an almost felsitic character, and is highly mineralised by arsenical and iron-pyrites. In Fig. 124, *a* is the intrusive dioritic mass; *b*, Silurian sedimentary rocks (contact schists); *c*, auriferous quartz veins; *d*, plane of contact.

Fig. 125 shows a few actual sections of veins in the Clunes district, Victoria.

FIG. 125



SECTIONS OF VEINS AT CLUNLS.

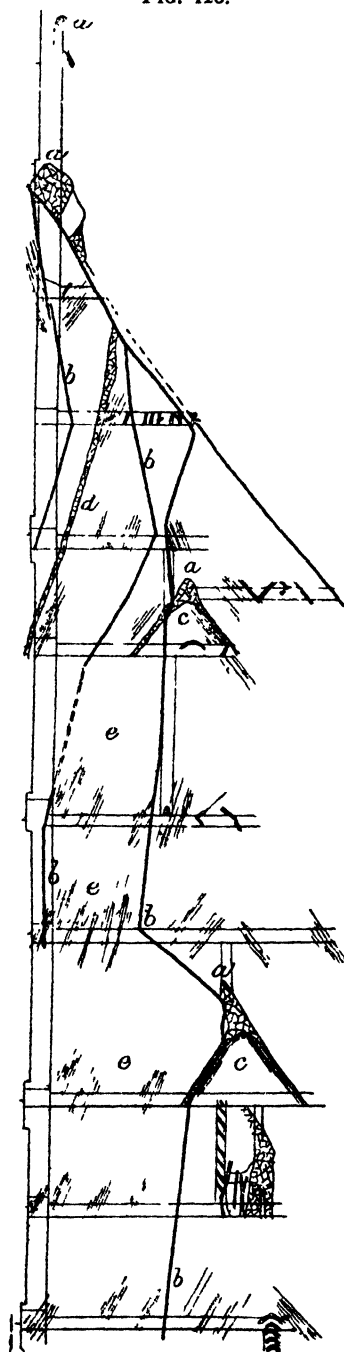
The Sandhurst gold-field, Victoria, is remarkable for the regular saddle formation of its quartz reefs. In the North Garden Gully United Co.'s main shaft, they underlie each other as shown in Fig. 126 (p. 367). They are curiously faulted by an intrusive "lava" (composed of dense crystalline basaltic rock) dyke, which follows irregularly along the strike of the veins, and intersects them at various depths. The reference letters indicate: *a*, caps of reefs; *b*, "lava" dykes; *c*, saddle reefs; *d*, west leg of reef; *e*, Silurian strata. This mine in nine years paid over 600,000*l*.

in dividends, and the market value is about 200,000*l.* It is closely resembled by the Great Extended Hustlers, which, in eight years, paid 560,000*l.* in dividends. The shaft has reached over 1400 ft. in depth without finding any productive reef below 733 ft.

Characteristics of Veins.—The veins of South Carolina are divided by Tuomey into two categories: (1) those in which the precious metal is found disseminated in the slates, or in beds between them, and called by the miners "slate mines"; and (2) those in which the lodes are true veins. In the King's Mountain range, the first mine towards the north on King's Creek is very remarkable, being a bed of rather poor iron ore (hematite) and common quartz, about 3 ft. thick, in mica-slate. The gold is confined to certain ferruginous portions, in which small quartz veins abound. At other places in this range, the gold is associated with iron as a yellow ochre. Above Easterwood Shoals, some of the mines consist of talco-micaeous slates, intersected by veins of felspar passing into kaolin, the auriferous veins being coarse crystalline granite; at Littlejohn's, not far distant, on the contrary, the gold-bearing lodes are quartz, and though granite veins are common in the mine, they contain no gold. Tuomey is "convinced that all the oxide of iron that occurs in the gold-veins was once sulphuret of iron, or pyrites, and that at a depth below the surface which is quite variable, the iron associated with the gold exists in that form."

The principal gold-producing mines of Colorado have been the subject of an exhaustive report by J. D. Hague. Concerning the lodes in general, he says they are all enclosed in a rock of one common type,

FIG. 126.



SADDLE REEFS.

chiefly granitic, with gneissic varieties. Their course, with few exceptions, is either due east and west, or north-east and south-west, the prevailing line along the prominent lodes being about 5° to 10° north of east. The dip is generally nearly vertical. The lodes possess the features of true fissure-veins, and are remarkably free from faults or displacements. The Bobtail lode has the reputation of having been the most productive vein of Colorado. The width of the vein ranges from a few inches, or a mere seam, to 10 or 15 ft., but does not exceed 3 to 4 ft. on an average. The enclosing rock is gneiss, sometimes of granitic character. The walls are usually well defined, especially the southern one. The vein matter is a quartzose and felspathic mixture, highly silicious, and carrying much free quartz, but not having the appearance of solid quartz veins; in many places, it has a granitic look, thus differing but little from the country-rock, and in such cases it is usually quite barren. The gangue is soft, whitish or pale-greenish rock, consisting chiefly of decomposed felspathic material, mixed with quartz, and thickly impregnated with iron- and copper-pyrites. The richer ore is concentrated in a seam of solid sulphurets, composed mainly of iron- and copper-pyrites, with less quantities of galena, zinc-blende, arsenical pyrites, &c. The gold is associated with the pyrites, most richly with the fine copper-pyrites, and varying in the iron-pyrites according to the closeness of grain. The productive part of the vein usually carries a seam of solid auriferous pyrites, 1 in. to 3 ft. wide, an average of 10 to 12 in. being deemed excellent; this seam is accompanied by a mass of gangue 1 to 4 ft. wide, with finely crystallised sulphurets generally disseminated through it. The last-named forms the bulk of the vein, and gives 6 to 7 oz. of retorted amalgam per cord (7 to 8 tons) of ore, the value being \$14 to \$16 (2*l.* 18*s.* 4*d.* to 3*l.* 6*s.* 8*d.*) a ton.

The auriferous region of Dakota is comprised within the Black Hills, and occupies a metamorphic range covering an area of 900 square miles, extending north and south through the central portions of the hills for a distance of 75 miles, with a breadth varying from 5 to 25 miles. The mineral resources of this region have been made the subject of a valuable report by W. P. Jenney, according to which the gold occurs in the following formations:—(a) In veins of quartz, more or less ferruginous, traversing the Archæan schists and slates. (b) In strata of slate, mineralised and altered by the action of waters depositing silica and iron-pyrites. (c) In the conglomerate forming the lower layer of the Potsdam sandstone, resulting in this case from the denudation of the ledges in the Archæan rocks. (d) In trachyte and porphyry intruded at the time of elevation of the Hills, in the interval between the Cretaceous and Tertiary periods. (e) In deposits in the slates and sedimentary rocks, produced by the intrusion of the trachyte and porphyry.

Nevada is especially remarkable for possessing the renowned Comstock lode. This has been described at tedious length by numbers of writers, but the summary of its formation has been succinctly stated by Clarence King in the following paragraph :—"The ancient Virginia Range, prior to the Tertiary period, was composed of sedimentary beds of the Great Cordillera system, which, in the late Jurassic epoch, had been folded up, forming one of the corrugations of that immense mountain structure which covers the western front of the American Continent. Accompanying this upheaval were outpourings of granite and syenite. The erosion which followed this mountain period escarped the ancient rocks, and modelled the eastern point of Mount Davidson into a comparatively smooth surface, whose average angle of slope sank to the east at almost 40°. In the late Tertiary, at the time of the volcanic era, the Virginia Range shared in the dynamical convulsions which gave vent to excessive volcanic outflows of immense volume and very remarkable character. The first, and, so far as the Comstock lode is concerned, the most important, was of propylite, or trachytic greenstone, which deluged the range from summit to base, covering large portions of its ancient surface, and leaving here and there isolated masses, which rose like islands above the wide fields of volcanic rock. Subsequently followed the period of the andesites, which, at their commencement, in the form of a thin intrusive dyke, penetrated a new-formed fissure on the contact plane of the ancient syenite and propylite. This earlier andesite period gave birth to the solfataras, which, bursting from a hundred vents, rapidly decomposed the surrounding rocks, and gradually filled the fissures of the Comstock with their remarkable charges of metal-bearing quartz. The latest flows of andesite poured out over the decomposed propylite, and since they are themselves unaltered, their appearance marks the period when solfataric action over wide areas had ceased. While it no longer maintained its energy through the broad zone of propylite, it still continued intensely active within the chambers of the Comstock lode. Metallic contents were introduced into the quartz, the clay-seams were formed by a rapid decomposition of the neighbouring propylite materials, the lodes reduced to a spongy semiplastic condition, and at last the final solidification of the quartz took place. Outside of the vein, two events of geological interest have occurred : first, the period of trachyte eruptions, when from the ruptures of the crust, parallel to the Comstock lode, vast volumes of sanidin-trachyte over-flowed the country ; and, second, the less powerful but still important outpouring of basaltic rock, which marked the close of the volcanic era. Within the vein, and probably caused by one or both of these latter volcanic disturbances, a pressure has been exerted which has crushed and ground the masses of quartz into minute fragments. It is interesting to observe

that, while this force was great enough to crush quartz-masses 150 ft. in breadth into mere angular pebbles, the disturbances were insufficient to cause any actual faulting of importance. Both within and without the vein, solfataras gradually came to a close. The heated currents of water, which even yet ascend into the lower levels of the mines, are evidence that at no very great depth a considerable temperature is still maintained ; but this is only a faint relic of a once intense action." Again, he remarks that "quartz forms the only gangue in the Comstock lode. Those small masses of carbonate of lime, which occur, intermingled with quartz, in the Gold Hill and Hale and Norcross lower levels, are rather to be considered as included mineral of accidental occurrence than as a true gangue. With the exception of small quantities of silver minerals contained in the clay sheets, where they are placed in contact with the bonanza, the whole silver-tenure of the lode is contained in the bodies of quartz. The ore itself is composed of native gold, native silver, silver-glance, stephanite, polybasite, rich galena, occasional pyrrargyrite, horn silver, and, with extreme rarity, sternbergite. Intimately associated with these, occur iron- and copper-pyrites and zinc-blende. Of these, pyrrargyrite and horn silver are rarities ; polybasite and sternbergite, in recognisable crystals, occupy a few scattered localities ; stephanite, in defined crystallisations, has been found in nearly every bonanza ; but the main body of the ore is a confused semi-crystallised association of native gold and silver, vitreous silver ore, rich galena, copper- and iron-pyrites, and zinc-blende."

All the Ruby Hill mines are on one large "bed-vein," which has been crushed, moved, and shattered to such an extent as to interrupt the ore sheet, if indeed it were ever quite continuous. The country limestone is more or less shaly and siliceous near the quartzite, but, in passing upward, compacts into thick heavy masses, whose original bedding is almost utterly lost, resulting in a massive capping, shattered by irregular jointed planes, and everywhere seamed and impregnated with veins and bunches of calc-spar. The ore-bodies occupy the shattered shaly limestone zone, bounded by the quartzite below, and the more massive limestone above. As must inevitably be the case with all similar deposits, the walls of this bed-vein are more or less irregular and undulating : the included ore-mass pinching or widening in accord with the accidents of pressure and metamorphism ; hence the remarkable changes in dip and thickness which characterise it. The foot-wall, wherever developed, is quite defined, more or less marked and streaked, with occasional clay salvages, which are never important or continuous. The hanging-wall is still less perfectly defined,—it is characterised by a broken surface, and by a zone of broken limestone blocks detached from the hanging country, and lying within the vein-material, surrounded by

bunches of ore and masses of oxidised ferruginous gangue. Between these walls, for an average thickness of 60 ft., lies the mass of ore, for the most part fairly uniform.

The Eozoic gneiss of the Brazilian coast-belt furnishes gold at numerous localities along its whole extent. Hartt thinks that the auriferous deposits occur in the upper part of the gneiss series. Gold is found in the gneisses of the Itacama Mountains. Gold also occurs in the gneiss of the Mucury basin, and in the north, as at Piancó. Over this whole region, the metal is found rather sparingly, and appears to be derived from the quartz veins traversing the gneiss; but probably the only instance where gold has been extracted from a quartz vein in the gneiss-belt is the mines on the Rio Bruscius, in Parahyba. In very numerous localities in the interior of the country, the newer metamorphic rocks are rich in gold. In the clay-slates, the quartz veins sometimes show free gold, with very little pyrites associated. The auriferous quartz varies much in character. Sometimes it is compact and milky, at others it is very granular and sugary. A specimen of vein-quartz with gold, from the mine of São Vicente, in Minas Geraes, is composed for the most part of clear, colourless quartz, in rather coarse granules, giving to the rock the appearance of a pure quartz sandstone, or of white lump-sugar (known in Brazil by the name of *cacó*); but in the same specimen, the quartz passes into a more compact bluish rock. In the quartz veins, as is elsewhere the case, the rock is not all auriferous, but the gold runs in streaks. The São Vicente specimen above described shows well-marked streaks, rich in free gold, which appear to have run parallel with the side of the lode. Where the vein-rock is rich in sulphides, the gold is, as a rule, not visible, but intimately mixed with the rock. The sulphides consist of magnetic iron-pyrites, which is the most abundant, and yields a little gold; common iron-pyrites is less abundant, and gives more gold; and mispickel or arsenical iron-pyrites, which is the principal gold-bearer.

Of the ferruginous deposits, none is more interesting than the so-called *jacutinga* formation, a pulverulent variety of itabirite, which is composed of micaceous specular iron ore, compact specular iron, rarely laminated, a little oxide of iron and quartz disseminated.

The so-called *guacas* of the United States of Colombia are very interesting objects for examination and scientific inquiry, on account of the novel geological conditions attending the presence of gold, which are exhibited by them. These *guacas* penetrate the hillsides at various elevations, differing greatly in their extent, and varying exceedingly in many instances from the horizontal direction. There are, however, several circumstances which are found associated with these excavations without exception, and some of them are undoubtedly the indications

which lead to the selection of the locality, and govern the course and extent of the *guaca*. At whatever elevation on the side of the hill the excavation may be, the opening is always at a considerable distance below the summit of the hill; and as the general direction of these tunnels is the horizontal, they penetrate the body of the hill at greater or less depths below the natural surface. The indication which governs the selection of the spot for commencing a *guaca* is the presence of a stratification called by the natives *peña*, and a detritus, either superimposed or partially mingled with the upper portion of this stratum, which yields gold. This deposit of auriferous detritus varies exceedingly in extent, but when it is confined to a few feet in breadth and thickness, it is styled a *venero*, or vein. The stratum called *peña* is felspar or gneiss, decomposed to a greater or less degree in its upper layers, which, coloured by iron sulphates, present different shades of blue. As one descends in these strata, they become more compact and hard; and in no instance is gold found below the extent of decomposition and consequent dissolution in these strata. The vein or auriferous deposit is composed of boulders of trap, greenstone, gneiss, and felspar; detritus of every description of rock belonging to this region, and decayed vegetation, such as trunks of trees and fruits, some of which have lost entirely their vegetable matter, and are impregnated with aluminous earths; while iron-pyrites, magnetic sand, and crystallised quartz, and lastly, by far the most important ingredient, gold, in large quantities, fill the interstices. Above the auriferous deposit is superimposed a layer of porphyroidal clay, of no great thickness generally, which is variously coloured by iron and its compounds. This layer is styled among the native miners here "*sepé*," and is not unfrequently absent, or of very limited thickness. Above this layer comes a stratification, which, from its position and character combined, presents an interesting geological phenomenon worthy of especial attention. The name given to this latter by the natives is *tosca*. It seems to be of volcanic origin, having been evidently superimposed upon the *sepé* when in a molten state. It has many of the characteristics of lava, and would seem to have been the result of volcanic action while the surface of this region was submerged below the level of the sea. This *tosca* is of varied thickness and extent, in some places reaching the surface, and frequently making a stratum which extends throughout the body of the hill, overlying the *sepé*. In some *guacas* it forms the roof for their entire extent; and its presence is considered one of the most favourable indications of the existence of gold in the *venero*. In most instances, the *tosca* is covered by diluvium and alluvial earth, to a greater or less thickness, forming the summits of the hills. Now and then, however, the *tosca* comes out to the surface on the plateaux of the *guaca* hills. This is the case on the

Cocuyos hill, one of the most celebrated of these eminences, where it may be seen and traced to considerable distances. The auriferous deposits in these *guaca* hills vary greatly in extent and disposition, as already remarked. Sometimes there is no distinct and uniform *venero*, but the deposit is distributed irregularly in layers of different thickness and limits, with interruptions like "faults" in a vein of quartz, composed of the layers of *peña* and *sepé* interposed in juxtaposition. This occurs generally when the deposit is located deep in the hill.

Gold occurs in Japan in small veins, traversing a porphyritic rock. The veins vary in thickness from 3 ft. to a mere sheet. The gold is associated with copper- and iron-pyrites, blende and galena. The thin veins contain the richest ore, while the larger veins are usually too poor to be worked. The large veins hold copper and lead; but neither are they rich enough in these metals to be worked. These large fissures are quite regular: run north and south, and dip 45° to 70° west. They are usually filled with decomposed porphyry, often in the form of a plastic clay. The copper-ore occurs in local deposits of limited extent, near the foot-wall. The large veins intersect and cut off the smaller gold-bearing veins, these latter being either branches of the main fissures, or faulted to a considerable extent by them.

With the exception of those of the auriferous locality near Portobello, the rocks of the Otago goldfield, New Zealand, consist throughout, according to Professor Ulrich's observations, and which are confirmed by Captain Hutton the Provincial Geologist's more extensive and detailed geological survey—of metamorphic schist, i. e. argillaceous mica-schist or phyllite, changing from the east towards the west into real mica-schist, with subordinate bands of chlorite-schist, or chloritic mica-schist. Only at two places within the extensive metamorphic district Ulrich obtained evidence of the existence of an intrusive rock, in small dyke-and-knob-like protrusions of a dark "hornstone-porphry." Massive occurrences of granite, such as characterise the neighbourhood of most of the Victorian goldfields, are quite absent. As a general rule, both phyllite and mica-schist, but more especially the latter, are rich in interlaminae of quartz, generally less than $\frac{1}{4}$ in. to near 1 in. thick, but sometimes assuming considerable dimensions—1 to 3 ft. in thickness—though with no regularity and permanency in strike and dip. These generally lenticular-shaped masses, and the small interlaminae of quartz, have at several places been found auriferous, which led to their being erroneously mistaken for true reefs or their leaders; and considerable amounts of money and labour have been wasted in their exploration. Gold at the rate of 6 dwt. per ton is said to have been crushed from this quartz: but as neither in the workings nor in the stuff worked, nor in the surrounding country, any indications of a reef are visible, the auriferous quartz found

must have been derived from the small interlamination, and the prospects would, therefore, certainly not warrant further expense in more extensive exploration. As regards strike and dip, the schists are subject to such great changes throughout the country as to render the taking of any mean a hopeless task. The auriferous reefs differ very much both in structure and mode of development.

Some reefs are true lodes, promising permanency in depth. They have well-defined foot- and hanging-walls and clay casings or selvages, and cross the country (phyllite) both in strike and dip, though the difference in angle is generally inconsiderable. In structure, development, and mode of occurrence, they resemble very closely a certain class of Victorian reefs, called "block reefs," the typical characteristic of which is, that they are composed of generally a number of blocks of quartz, either with contractions of the lode-fissures between, or, which is the case here, alternating with blocks of "mullock"—the latter term meaning gangue matter, consisting of country-rock slipped into the lode-fissure, where, in course of time, it became more or less mineralised, impregnated with pyrites, and traversed by small quartz veins. These quartz and mullock blocks, which reach sometimes considerable dimensions, extend hardly ever vertically downwards, but show an endlong dip in strike within the fissures, in the same reef invariably, in the reefs of the same district generally, in the same direction. The feature may, in fact, be considered as an oblique banded structure on the large scale.

The thickness of the blocks reaches in some cases 12 ft., and their dip in strike is generally at rather flat angles. Gold occurs both in the quartz and mullock blocks; but the former have hitherto invariably been found the richest, and in them its distribution is mostly not uniform throughout, but a rich small shoot dips at a flat angle across the block; or, a number of narrow, richer runs, dipping in the line of the blocks, are separated by poorer quartz: or, a certain thickness on the hanging- and foot-walls is richer than the centre, &c. The appearance of the quartz varies very much according to whether it comes from the surface or from beneath the water-level. In the former case, it is white, opaque, and mottled and striped with brown iron-ore, or ferruginous slaty matter; in the latter, it shows a bluish colour, is slightly translucent and glassy, and full of blackish spots and seams of slaty matter, which, as well as the quartz itself, are more or less densely impregnated with pyrites. The reefs are rather poor in gold, the yields ranging not much above 5 dwt. per ton throughout. Still, considering the thickness and extent of the quartz blocks, the facility with which they can generally be worked, and that sometimes the intervening mullock is worth crushing, on account of thin rich quartz seams traversing it, they ought to be

profitable to work for at least several hundred feet in depth ; more especially if attention is paid to the saving and treatment of the pyrites.

The thickness of the Bendigo reefs is, on the average, but small, being generally from $\frac{1}{2}$ to 2 ft., and rarely exceeding 4 ft. As far as workings have proved, they do not consist of solid quartz throughout, and but seldom so for any considerable extent in strike and dip. They represent, in fact, in certain respects, "block reefs," though with this difference, that blocks of quartz and mullock of irregular size and outline are more or less irregularly intermixed, and do not dip at certain angles, and in the same direction in strike. Most of the reefs of Bendigo show well-defined walls with clay casings, strike nearly uniformly east and west, are mostly traceable for long distances, and—what constitutes them very "strong" in a mining sense, and indicates permanency in depth—they traverse horizontal or very flat-bedded mica-schist vertically, or at very steep angles. Not uncommonly the reefs split in strike into branches, which, though deviating at first from, gradually assume again the strike of the main reef, and run thus pretty close and parallel together, some increasing to the same or even a greater thickness. Besides "leaders" that dip towards them, so-called "droppers," dipping at generally flat angles away from them, have also been observed in some of the reefs. With regard to the mode of occurrence of the gold, it would seem that the metal is accumulated in rich shoots of variable width, that dip at rather steep angles in strike in the quartz blocks ; whilst the portions intervening between the shoots, including the mullock patches, are poor, but generally rich enough, or of such limited extent as to render it most economical to work them with the rest, without resorting to special selection. The yields have been by no means low on the average, as they ranged up to 2 oz. per ton, and from hardly a single reef were they reported much below $\frac{1}{2}$ oz. per ton. The prospects of the reefs in depth are favourable, both as concerns persistency in auriferous character and regularity in average size. But in speculating upon profits to be derived from future working, several important points must not be left unconsidered—namely, the comparatively small size of the reefs, expenses connected with getting rid of the water, and greater difficulty in extracting the gold from the quartz ; for below the water-level, which lies in several of the reefs considerably higher than one would suppose from their elevation above the nearest permanent surface water, the seamy quartz is throughout very metalliferous ; in fact, it promises at greater depth to become more highly charged with metallic sulphides (arsenical and iron-pyrites, galena, zinc-blende, &c.) The expense of working narrow reefs will, in depth, also considerably increase, in consequence of the increasing hardness and closeness of the country rock, which latter is comparatively more unfavourable to work on account of its horizontal

bedding. However, against this may be placed, as a perhaps more than adequate set-off, the high dip of the reefs, which much facilitates working, and what is of greater importance, that the hardness of the rock in conjunction with its horizontal bedding renders the supporting of the workings very cheap and simple.

The generality of the reefs of the Carrick range are peculiar clayey ferruginous "mullock reefs" or rather "quartz-mullock reefs," so soft that they can mostly be worked by pick, without the aid of boring and blasting; and the quartz, which apparently forms no large percentage of their mass, occurs only in the shape of coarse sand and small angular and slightly rounded pieces—such reaching or surpassing the size of a fist being rather rare. Whether it represented originally interlaminae in the mullock, or was formed in veins, is uncertain, but a kind of banded structure in the line of dip of the reefs speaks in favour of the latter. These reefs vary in thickness from less than 1 ft. to over 6 ft.; they strike in all directions across the country, but are only of short extent, and differ very much both in direction and angle of underlie, the latter ranging from vertical to less than 20°. Some of the reefs show also much irregularity in their course, for they expand and contract, twist and curve in strike and dip in quite a peculiar manner, and are frequently faulted by slides and cross-courses, so that it requires very great attention and perseverance not to lose them. Considering all these points in connection with the fact that the country rock (a rather soft phyllite) is also very much disturbed, both in strike and dip, it appears next to certain that not only the peculiar soft and gravelly nature of the reefs, but also the exceptionally flat dips of some, are not original, but due to strong disturbances, caused most likely by the intrusion of dark hornstone-porphyry. Unfavourable as these features no doubt appear, touching straightforward and uninterrupted working of the reefs in future, Ulrich feels no apprehension of the latter giving out suddenly or at a limited depth, for they are in every respect true lodes, crossing the country both in strike and dip, and showing most frequently the hanging-wall, less frequently the foot-wall, well defined, and separated from the mass by clayey casings, mostly polished and striated, representing the so-called "slicken-sides," which afford unmistakable proof of movements of the walls of the reefs. The gold, both in quartz and mullock, is very fine, and owing to the soft and ferruginous nature of the stuff, specks can rarely be seen during working. It occurs chiefly in shoots dipping in strike, less in irregular patches, but also pretty generally distributed throughout the whole extent of the reefs, as far as opened. The yields of most have in the average been very fair, as they ranged between $\frac{1}{2}$ and $1\frac{1}{2}$ oz. per ton. On account of its softness, the quartz mullock is easily crushed, but the saving of the fine gold requires great attention,

and the supply of water, which the proper treatment of this kind of stuff requires, is rather above the average. There is not much pyrites observable in the reefs; still the ferruginous character of the mullock, as being no doubt a result of its decomposition, points to its former presence in larger quantities, and it may with certainty be expected to increase in abundance in depth. As regards the expense of working the reef, the soft nature of both the mullock and country rock renders it small in respect of exploitation proper, but rather large on account of timbering, and the cost in this respect increases of course with the flatness of dip of the reefs.

The reefs of the Arrow and Skipper's Creek group are true massive lodes, ranging from 4 to over 20 ft. in thickness, which cut through the country both in strike and dip (the latter being generally steep), and show more or less well-defined walls, with clay casings; a number are traceable for long distances—some for miles—in strike. In point of composition and structure, they approach, however, far more mullock reefs than true quartz reefs, representing fissures partly filled with country-rock *débris* full of interlaminated quartz, partly by bunches and veins, of variable size, of true reef-quartz. The mullock in the larger reefs is generally predominating, and forms, in places where the width much increases, by far the greater part of the mass. In fact, the thicker a reef is, or the wider it becomes, the more mullock it contains, whilst decreasing thickness is connected with a relative increase in quartz, and the reefs become also better defined. The bunches and veins of reef-quartz occur either on the hanging- or foot-walls, or on both walls, rarely in the centre. They appear to dip shoot-like in strike, and are generally payably or richly auriferous; but the somewhat mineralised mullock, with its interlaminations and fine cross-veins of quartz, has also been found to contain gold throughout, though generally only in payable quantity within the line of the quartz shoots, or where the reefs much contract in size. The yields have ranged from several dwt. to over 4 oz. of gold per ton, but the average is 10 to 16 dwt. per ton. Although not yet opened below permanent water-level, both quartz and mullock are already highly charged with pyrites, which seriously interferes with the satisfactory saving of the generally fine gold during crushing. The country is, with regard to most reefs, very favourable for their being easily and cheaply worked, owing to the highly precipitous nature of the mountains which they traverse, and the deep valleys and gorges from which they can be opened, either directly in strike, or by but short cross adits. Still there is one serious drawback to regular and systematic working, affecting the most important of the reefs within certain depths, namely, although the country-rock (mica-schist) appears on the large scale but little disturbed in strike and dip, it is throughout highly fissile, and traversed by

numerous cracks and joints ; and these features combined, aided by percolation and freezing of water, have originated enormous slips from the precipitous mountain sides—faults, in a certain sense—by which the continuity of the reefs is completely broken. These disturbances, which render working dangerous, and cause much expense in timbering, extend however, to within the level of the nearest gorges or valleys only ; below this there is every probability of the lie of the reefs becoming less broken, though the existence of smaller faults and other irregularities must there also be apprehended, on account of the fissured nature of the country. As regards continuance of the gold in depth, Ulrich sees no reason to doubt it, yet thinks it very likely that the increasing abundance of pyrites may, as in Victoria, be connected with a corresponding decrease of fine, free gold, in depth.

The reefs of Macrae's Flat and Shag Valley represent either so-called "layer-lodes" or are merely interlamination between the beds of phyllite that form the country rock. Touching the layer-lodes, their general characteristics are, that they strike and dip with the country, having the foot-wall (one and the same country) generally pretty smooth throughout, but the hanging-wall mostly irregular, uneven, and traversed by leaders. They are, on account of this mode of relation to the country, subject to all the changes in strike and dip of the latter, and, if these are great, are liable to frequent alterations in thickness, and can generally not be depended on for persistency in depth. Their dip is rather flat (less than 45°), and they are 2 to 5 ft. thick on the average, being composed of mullock and rather good-looking quartz ; the latter generally predominating and occurring in bunches and veins, mostly on the foot-wall, sometimes on the hanging-wall, or being irregularly intermixed with the mullock. The gold occurs both in the quartz and the mullock, though mainly in the former, and the yields have ranged from a few dwt. up to 2 oz. per ton. The reefs show a good thickness for considerable extent. With regard to the interlamination, which were considered to be reefs, and of which some are in places several feet thick, they are quite unreliable concerning gold-bearing character and extent in strike and dip, though several carried good gold (above 1 oz. per ton) near the surface.

In the steeper hills bordering the Thames valley, New Zealand, gold has been found. They are composed of trachytic tufa, containing large rounded boulders of dolerite and greenstone. Behind, again, the ranges are still higher, and are probably composed of blue and pink slates, for boulders of these rocks are seen in the stream. Mining claims are held in decomposing ferruginous tufa, containing boulders of dolerite and dark-green greenstone, or melaphyre. The general direction of the veins is north-east and north-north-east, but they appear to be highly irregular, and to break up near the boulders. The gold is about 16

carats fine. The height of the mines above the sea-level varies from a few feet to about 1800 ft. ; and the present sea-level is more than 100 ft. above what it has been in, geologically, quite recent times.

The base of the tufa formation is of varying height, as it rests upon the old denuded surface of the slates. Many of the lodes are of a most permanent character ; and in those claims where only small irregular veins occur, they are so numerous that when one is lost another is almost sure to be found, and so, in Hutton's opinion, there is every probability of all the mines extending to the base of the tufa formation. Whether they will or not pass into the underlying slates remains to be proved ; but in one case at least this is so, and it is therefore probable that the other large reefs and lodes will do the same ; but not, perhaps, the smaller branching veins, which are the effects only of the contraction of the tufa. The few auriferous lodes found as yet in the slates, in comparison with the number found in the tufa, is no argument against their penetrating both formations, for the exposed surface of the tufa is so much larger, and has been so much better explored than that of the slates, that the result could not well have been otherwise. According to E. H. Davis, the presence of gold in the reefs in payable quantities is not marked by any distinct features, the manager of each claim having a different reason for expecting a reef to turn out well or the contrary. Thus : (1) in Hunt's and the Kuranui mines, gold is expected when the reef becomes rich in pyrites and the quartz more or less dislocated ; (2) in the Long Drive, gold is expected when the reef besides having the ordinary grey flucan, contains patches of a shiny-black greasy compound, which stains the hands a permanent black ; (3) in the Golden Crown, it is considered a good sign when the reef contains pyrites, is a dull-grey colour, and is more or less mullocky ; (4) Waiotahi,—quartz not too hard and compact grey flucan and pyrites ; (5) Dauntless and Sink to Rise,—reef slightly decomposed, with small leaders of quartz running through it, the whole being a dirty white. The managers are all agreed on one point, viz., that they cannot give a satisfactory reason for expecting gold in the reefs.

Davis's own personal observations led him to the following conclusions : (1) Pyrites in the matrix of the lodes is a *sine quâ non* ; (2) the hanging-wall or roof need not of necessity carry pyrites, but rather the richness of the reef is increased by the diminution of pyrites in the roof, provided the foot-wall is rich in them ; (3) leaders joining the reef on the hanging-wall probably increase the yield of gold for a time : leaders from the foot-wall, seldom ; (4) the direction of the leaders in the tufa-sandstone has nothing whatever to do with the auriferous quality, provided the leader runs more or less parallel with the direction of the diorite-sandstone ; (5) the free gold will decrease with the depth of the workings, but if the land has been elevated at some previous time for a

considerable period above its present level, and consequently the water-level lower, the gold will probably continue down to that depth, but not beyond it : the claims near the beach are already beginning to find the reefs harder, the pyrites more solid, and the yield of gold less ; (6) the leaders nearly all being formed by segregation, are particularly amenable to atmospheric agency in liberating the gold.

The rich goldfield of Charters Towers, Queensland, occupies the western edge of an area of granite and syenite, bordering on the little-altered slates and grits of the Sandy Creek district. The reefs form a sort of horseshoe bend, with its convex side to the south, and underlie towards the centre of this curve at a low angle. At present, visible gold is a rarity in the reefs ; but experience having shown what classes of stone yield most when crushed, the miner is in no way dispirited even if he never sees the "colour" in his workings. There is considerable monotony in the underground operations, which present the same country, the same gangue, and the same phenomena at and above the water-level. The surface of a reef is sometimes exceptionally rich in gold. The region of "brownstone" (pyrrhotite and peroxide of iron) coincides with that superficial part of the rock which is not permanently saturated with water, and into which the atmosphere penetrates. A second region, that of "mundic" (auriferous sulphides), is co-extensive, so far as is yet known, with the zone of permanent saturation. The rainfall is understood to circulate "chiefly within a film of the rocky crust not much exceeding 2000 or 2500 ft." (Dr. Page). As, however, the lower limit of saturation must vary according to the compactness of the rock and the weight of the water, it may be accepted as certain that this is an over-estimate for a district of small rainfalls and hard granite rocks. In the progress of denudation, the upper zone, which may be called the "zone of intermittent saturation," must necessarily invade the lower zone, that of "permanent saturation," while the lower limit of the latter will sink deeper and deeper into the crust of the earth. The brownstone is mundic which has undergone oxidation, the pyrites being altered to pyrrhotite. As, however, there is no reason to doubt that the gold exists in the mundic stone, it seems difficult to account for its greater abundance in the superficial brownstone. The explanation is this: On the denudation and oxidation of the upper portion of a reef, the gold mixed with the denuded sulphides is set free, and is carried with the aid of rains and by virtue of its own high specific gravity into the porous weathered upper part of the reef, which is next in turn to undergo denudation ; and it may thus happen that the "capping" of a reef is enriched with loose gold. Add to this that the working of the surface is much easier than that of the deep levels, and it will be seen how, in many cases, only the upper part of a reef may be payable. Gold disseminated through the granite

rocks themselves is occasionally evident to the naked eye, but it may be present in many cases in quantities so small that neither the microscope nor ordinary analysis will detect it. That it has been collected in veins from the granite rocks, by means of lixiviation, is rendered very probable from the fact of the auriferous pyrites occurring in the "zone of permanent saturation." Arsenic is scarcely known in any form in the field.

The Sterling Reef, South Australia, is in reality not a genuine quartz reef, but conforms more to what are called mullock reefs in Victoria. One may compare it to a large fissure, in places 30 to 40 ft. wide, filled confusedly with larger and smaller masses of the wall-rocks—grit and slate—and the interstices between these masses occupied by mullock, enclosing irregular pockets and veins of quartz, through which (but more especially through the quartz) rather solid crystalline specks of gold are distributed. Iron-pyrites has not as yet been observed. The mullocky portions (irregular veins, as it were) cross the reef-mass in all directions (horizontally, vertically, and obliquely), and on this account, as each vein cannot be specially followed, the workings have to be carried on by quarrying the reef out *en masse*, and classifying the stuff afterwards,—a rather troublesome work, during which it cannot be prevented that a large amount of poor or non-auriferous matter becomes mixed with the quartzose mullock; hence the low yield of the latter of only 2 to $2\frac{1}{2}$ dwt. per ton on the average.

The Victorian reefs have been closely studied by Nicholas, who found that of 998 reefs, 841 are situated in the Lower, and 157 in the Upper Silurian rocks. These are subdivided into reefs bearing west of north and east of north. Of those in the first-named class of rocks, 554 had an average strike of N. 20° W., and 117 in the latter had an average strike of N. 34° W.; whilst there were 287 reefs in the Lower Silurian that possessed a strike east of north, averaging N. 11° E., and 40 reefs in the Upper which had a strike of N. 27° E. From these figures, it will be seen that out of a total number of 998 veins in both formations, 671 had a direction west of north, and only 327 ran east of north. The variation of these reefs to west and east of the true north point in the Lower Silurian was just one-half that in the Upper Silurian rocks; or, to put it in other words, in the first-named rocks, the average strike of the reefs west and east of north added together makes but 31° , whilst that of those in the second makes 61° . It appears then that the auriferous veins in the Lower Silurian formation are much more nearly north and south than those in the Upper. As a rule, the auriferous veins run parallel with the strata in which they are enclosed, and the greater number of the most extensively wrought and richest veins in Victoria strike west of north. The reefs are naturally divided into

seven groups, of which two are important and five of less importance. The two main groups or belts are well defined, have a general direction north and south, and are in the Lower Silurian strata. The eastern one contains 774 distinct quartz reefs actually proved to be auriferous. In the western main group or belt there are 535 known auriferous reefs. These two great belts are separated by the 144th meridian of longitude on the Map of Victoria, and physically by a barren and depressed strip of strata 15 miles in width. Of the five groups of lesser importance, two lie in the Upper Silurian and three in the Lower Silurian rocks. In the first group are 530 distinct auriferous reefs. The second group in the Upper Silurian contains 323 auriferous reefs, and the course of the group is north-west. In these two lesser groups, many considerable tracts of Upper Silurian strata intervene between the gold-fields, and are either deficient in auriferous reefs, or the reefs in them are not proved. The third group to the east is in what is believed to be Lower Silurian rocks, and contains 893 known auriferous reefs, striking west of north.

In Victoria, the veins run at right angles to the main dividing range ; but parallel with the principal spurs or tributary ranges of the main divide ; and the general direction of these ranges or spurs on the gold-fields is north or north-west at about right angles to the main dividing range. Auriferous cross-reefs occur in the Lower Silurian formation to the number of 61, of which 53 have an average strike of N. 60° W., and 8 have an average strike of N. 74° E. In the Upper Silurian, cross-reefs are found to the number of 24, of which 15 have an average strike of N. 85° W., and 9 of N. 71° E. Of the dips of auriferous veins in 134 instances, 78 in the Lower Silurian had an average of 61° W., and 22 of 62° E. ; and 19 in the Upper Silurian formation averaged 72° W., and 15 had a mean dip of 49° E. Out of the 134, 97 dipped to the west, and 37 to the east. In addition, Nicholas got the direction (but not the angle) of the dip of 41 other reefs situated in the Lower Silurian, 33 of which underlay to the west, and 8 to the east ; and also found records of 16 vertical reefs. It does not appear that the richness or poverty of auriferous reefs is affected by the variation of the angle of underlie. The greater number of the reefs seem, however, to have a dip of about 60°. The majority of the most important reefs in Victoria appear to have a western underlie, although there are many rich reefs that dip east. Nicholas also mentions that the saddle-formed reefs so characteristic of the master lodes of the Sandhurst gold-field are not confined to that field.

Respecting the shoot of gold, Nicholas got particulars of the dip of 57 "pay chimneys" (as they are called in California) or shoots, 25 of which dipped to the north, 28 to the south, and 4 were vertical ; the prevailing angle of the north and south shoots was between 25° and 50°. There

may be some difficulty in defining the dip of the shoot in reefs that contain fine gold evenly distributed ; but in all cases, if a longitudinal or working section were kept, the course of the vein worked out would show the dip of the shoot of quartz, with which the shoot of gold usually conforms. In Nicholas's own experience, he has found that the shoot of quartz is invariably at *right angles* to the lines of striation on the walls or laminations of reefs, or to the heads or joints in the bounding rocks. In California, it has been observed that the pay-chimneys run *with* the lines of striation. That the dip of the shoot of quartz or gold should be known, is a matter of the first importance in the working of all mines, and especially to adjoining mines.

Nicholas gathered the widths of 286 Victorian reefs. Of these, 237 were in Lower Silurian rocks, and 49 in Upper Silurian. He divided these widths into three classes, viz. (1) under 5 ft., (2) over 5 ft. and under 10 ft., (3) over 10 ft. Of the 237 reefs in the Lower Silurian, 168 were under 5 ft., and averaged 2 ft. 5 in. ; 45 were over 5 ft., and averaged 6 ft. 8 in. ; and 24 were over 10 ft., and averaged 29 ft. Of the 49 reefs in the Upper Silurian, 44 were under 5 ft., and averaged 2 ft. 1 in. ; 1 was over 5 ft. (or 8 ft. 5 in.) ; and 4 were over 10 ft., and averaged 15 ft. 6 in. From the above figures, it will be seen that the reefs in the Lower Silurian are much wider than those in the Upper Silurian formation, and it is found that the thickest reefs have proved the richest and most extensively worked in Victoria.

In the catalogue of auriferous quartz specimens in the collection of the Victorian Survey department, of 108 of the most important reefs from which fair samples were obtained, 59 had a laminated structure. The same data show that, of the 108 reefs, 48 contained gold in cavities ; 9 specimens only were found to contain gold in the solid quartz ; and in the laminations, gold was observed to occur in 24 reefs. These facts point out that the greater part of the free gold in the Victorian reefs is not enveloped in solid quartz, but occurs in a comparatively loose state in the cavities and laminations, with clay-slate, galena, pyrites, and zinc-blende : from which an inference may be drawn, that the freer the tailings run off from a quartz-crushing mill are from any other substance than quartz, the less is the loss of gold.

The basaltic dykes in the Lower Silurian of Victoria appear to conform in strike [direction] with the auriferous veins with which they are associated ; they cut through the veins and strata at all angles ; have a tortuous winding course ; occasionally, when they have been intercepted by a reef, they follow its underlie for short distances ; or they are found more frequently to fault the reefs ; in width, they are irregular, varying from 1 in. to 9 ft. 6 in. ; and have been found to split into two or more streams. Some dykes consist of compact hard basalt containing olivine,

and it would be difficult, if not impossible, to distinguish the rock of which they are composed from the newer basalts which are spread over such large areas in Victoria; whilst others are decomposed into white, yellow, grey, and speckled brownish clays. By miners, they are now considered certain guides to reefs, and Nicholas pointed this out as far back as 1863. If dykes can be said to have a dip, it is found, in working reefs, most frequently to be opposite to that of the reefs they intersect; but their general direction will no doubt prove to be vertical. These dykes record, by lines of striation and polished faces, movements on either side of them in the strata.

Of auriferous lodes or reefs in other countries, it is observed that the strike most common is north and south, and that there are a half-dozen mines in which the lodes may be classed as east and west or cross-reefs. It is stated, however, that the veins in Minas Geraes, Brazil, strike most frequently east and west. In Northern Mexico, and in Virginia, Orange, and Buckingham counties, United States of America, the strike of the veins deviates to the east of north, whilst those in California, British North America, and Oregon, U.S.A., strike either north and south, or deviate to the west. The east and west lodes are generally poor when compared with the north and south. The fact noted by Henwood, that the veins in Brazil run parallel with the mountain chains, has also been observed to be true in California. The lodes dip both east and west, and the average dip of those of which the angle is given is 55°.

Of 28 veins, 8 under 5 ft. averaged 2 ft. 9 in.; 3 over 5 ft. averaged 7 ft.; and 17 over 10 ft. averaged 27 ft. In common with Victorian reefs, these lodes or veins are proved to be ever varying in thickness; to have rich gold-bearing shoots or pay-chimneys, and poor zones, which succeed one another vertically or horizontally; to be laminated, have slickensides, dykes, and slides; to be nipped out—the walls of the veins coming together; and to possess the same mineral composition, consisting most commonly of quartz, arsenical and iron-pyrites, galena, zinc-blende, copper-pyrites, silver, antimony, and calcspar. With regard to the last-named mineral, in Nicholas's opinion, although it has hitherto only rarely been found in auriferous veins in Victoria, and then mostly in small quantities, it will be found to occur commonly and in quantity as the reefs are mined to greater depths. The calcite observed in the quartz from the Magdala mine and Victoria reef occurred in thin veins in compact quartz, and was only discovered after careful examination; in some of the other cases mentioned, it was found in cavities. In the "lava dykes," it sometimes occurs in thick seams. Whether the occurrence of this mineral in quantity in the Victorian reefs will have any effect on their auriferous character, is a matter of speculation; but it is not likely that the reefs will be richer for its presence. It is, however, probable that the

reefs will contain more silver, copper, lead-ore, and pyrites. Calcspar has not yet been found to exist to any large extent in the reefs down to 1000 ft.

The oft-repeated dictum, that quartz-veins which have proved very rich in the surface decomposed portions of a dyke, will become poorer as the rock becomes hard and compact with greater depth, does not always hold good. In the Wood's Point mines, some of the richest stone was obtained from quartz floors traversing exceedingly hard and undecomposed dyke stone.

EXTRACTION OF THE MINERAL.

Obviously the operations entailed in winning the auriferous material from the bowels of the earth are similar in most respects to those adopted in any other branch of mining. The veins have to be found by trial borings, and reached by tunnels and shafts ; the rock has to be broken down, and then drawn out of the mine for further treatment. These operations constitute the mining proper and belong to the science of mine engineering. As such, they do not claim lengthened treatment in the present volume, which deals with gold mining in particular rather than with mining in general. Nevertheless, there are a few points on which it may be advisable to dwell briefly.

Boring.—One of the foremost is the use of drilling-tools, both with the object of examining the nature of the ground proposed to be mined before incurring the expense of driving shafts and tunnels, and also for the purpose of introducing explosive compounds for breaking down the rock mass in the mine workings.

The old-fashioned systems of boring by pulverising the rock have almost universally given place to the diamond drill, which cuts its way, and permits a solid core to be drawn out of the hole, representing exactly the strata bored through.

The general principle of boring with the diamond drill is the same, the different machines, by comparatively slight changes, being applicable to any kind of rock-drilling. For deep boring, or prospecting mineral lands, a machine is used with a double oscillating cylinder engine, mounted on an upright, or horizontal, tubular boiler. The capacity of the engine varies according to the depth and size of hole requiring to be bored. These machines have a screw shaft made of heavy hydraulic tubing, 5 to 7 ft. in length, with a deep screw cut on the outside. The shaft also carries a spline, by which it is feathered to the lower sleeve gear. This gear is double, and connects by its upper teeth with a bevelled driving gear and by its lower teeth with a release gear, which is a frictional gear, and is fitted to the lower end of the feed shaft, to

the top of which a gear is feathered, fitting to the upper gear on the screw shaft, which has one or more teeth less than the upper gear on the feed shaft, whereby a differential feed is produced. This frictional gear is attached to bottom of feed shaft by a friction nut, thus producing a combined differential and frictional feed, which renders the drill perfectly sensitive to the character of the rock through which it is passing, and maintaining a uniform pressure upon the same. The severe and sudden strain upon the cutting points incidental to drilling through soft into hard rock with a positive feed is thus avoided. The drill rod, made of heavy lap-weld tubing, passes through the screw shaft, and is held firm by a chuck at the bottom of the screw shaft. To the lower end of this tubular boring rod the bit is screwed, and to the upper end a water swivel, to which connection is made with the steam pump. By means of this pump a constant stream of water is forced down through the hollow drill rod, thereby keeping the bit cool and the hole bored clear of sediment, which is forced by the water-pressure up the outside of the rods to the surface. The hollow bit is a steel thimble, having three rows of diamonds (bort or carbon) imbedded therein, so that the edges of those in one row project from its face, while the edges of those in the other two rows project from the outer and inner periphery respectively. The diamonds set in the first-mentioned row cut the path of the drill in its forward progress, while those embedded upon the outer and inner periphery of the tool enlarge the cavity around the same, and permit the free ingress and egress of the water as above described.

The screw shaft, being rotated and fed forward, rotates the drill-rod and bit; and as the bit passes into the rock, cutting an annular channel, that portion of the stone encircled by this channel is of course undisturbed; the core barrel passing down over it preserves it intact until the rods are withdrawn, when the solid cylinder thus formed is brought up with them, the core-lifter breaking in at the bottom of the hole, and securely wedging and holding it in the core barrel. Where a core is not required, the perforated boring-head can be used, the detritus being washed out by the water introduced through the drill rod the same as when boring with the hollow bit. In order to run the screw shaft back after it has been fed forward its full length, it is only necessary to release the chuck and to loosen the nut on the frictional gear, which allows the gear to run loose; then the screw shaft will run up with the same motion which carried it down, but with a velocity sixty times greater; that is, the speed with which the screw shaft feeds up is to the speed with which it fed the drill down as sixty to one, the revolving velocity in both cases being the same. By tightening up the chuck and nut on the frictional gear, the drill is ready for another run. The drill-rods may be extended to any desired length by simply adding fresh

pieces of tubing, the successive lengths being quickly coupled together by an inside shoulder nipple coupling, made of the best forged iron, and having a hole bored through the centre to permit the passage of the water. In order to withdraw the drill rods, they are uncoupled below the chuck; the swivel head, which is hinged, is unbolted and swung back, thereby moving the screw shaft to one side, and affording a clearance for the rods to be raised by the hoisting gear on the machine without moving the drill. By the erection of a derrick of sufficient height, it will be necessary to break joints only once in every 40 to 50 ft.

The advantages claimed for diamond drills over the steel or percussive system of drilling are:

(1) They drill rock faster than is possible in any other way; not only boring more feet in any given time—as a day or an hour—but accomplishing far greater results in the aggregate of a month's or a year's practical use.

(2) They perform a given amount of work more cheaply than it can otherwise be done, saving at least one-third the entire cost of heavy excavations (including the blasting and removal of material) as compared with hand labour, besides economising time in a much larger ratio.

(3) They are extremely simple, both in construction and operation, and seldom need repairs, the very best material and workmanship being used in their construction; and workmen of ordinary intelligence are perfectly competent to operate with them successfully. These machines, it will be remembered, are not subject to the constant and destructive shocks of concussion against the rock, which disable the best percussion machines so often, and render them so expensive to keep in repair, nor have they any delicate parts or nice adjustment to be carefully watched. Every part is equally simple and durable. The diamond teeth are the only part of the tool which comes in contact with the rock, and their hardness is such that more than 2000 ft. have been drilled by the same points with but little apparent wear. The cost of resetting the diamonds so as to present new points, is very slight, and no special skill is required for the operation. Other repairs are seldom needed.

(4) They produce holes uniform in diameter—not three-cornered or funnel-shaped, as must necessarily result from percussion drilling, but perfectly cylindrical from top to bottom—a feature of great importance in blasting, as the force of the explosive material is thereby fully utilised, and its practical effects are greatly increased. They are not deflected from a right line by seams and crevices, nor impeded in their progress by the hardest rock.

(5) The great advantage of being able to bore blast holes to the "bottom of grade" is well understood by rock contractors. For grades

of over 20 ft. in depth, this is the only machine with which holes can be readily bored. They bore as rapidly and cheaply at a depth of 50 to 100 ft. as at the surface, while at a depth of 500 to 600 ft. there is but little appreciable difference. With the prospecting machines, thousands of holes, from 300 to 1500 ft. in depth (many of them horizontal), have been bored through solid rock.

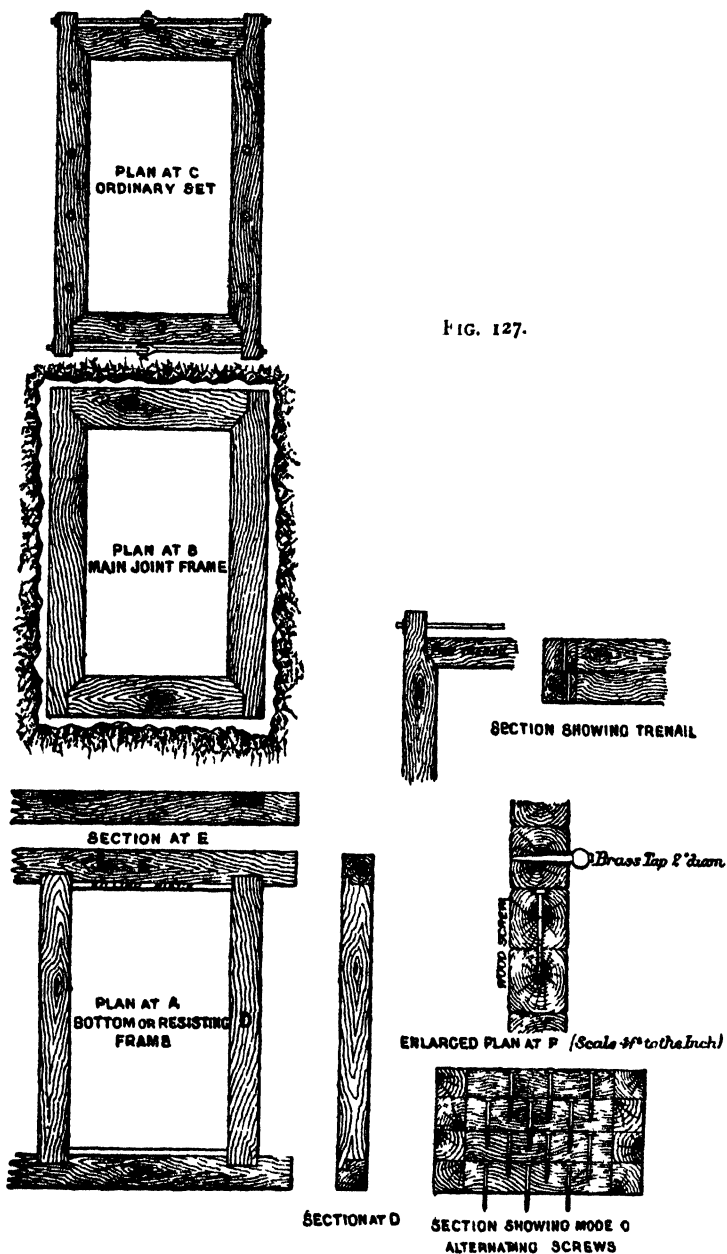
(6) All the drills are adapted to bore holes at every angle of either a vertical or horizontal circle, and in a shaft or tunnel they bore as close to either side-wall as a hand-drill can be turned. The peculiar shape of the boring-bit, holding a cylinder of solid rock inside, prevents the drill from running out of line; hence the hole bored, however deep it may be, is perfectly straight.

(7) They are not only adapted to shafting, tunnelling, well-boring, submarine blasting, and all kinds of rock-excavating in mines, quarries, railroads, &c., but in their application to "prospecting" they accomplish most important results, otherwise wholly unattainable. By their use only can mines be penetrated to a depth of 1000 ft. and upwards through solid rock, vertically or horizontally, and perfect samples of mineral taken out the entire distance, disclosing the character and value of the mine by means of a single drill-hole. It should also be observed that the samples so obtained are not disintegrated fragments of rock, but continuous solid cylinders, showing clearly the stratification and character of the ground so prospected.

Wherever these drills are used, it is necessary to arrange for a regular supply of diamonds. The two kinds of diamond used in drills are known as "bort" and "carbons." The bort is a real diamond, which, owing to imperfections, is useless as a gem; being nearly globular in shape, it is generally set in the outer edge of the drill, as being less likely to catch in irregularities of surface in the rock. The carbon is a black stone of very varying shape and usually sharp-edged. The bort is much the harder, and resists greater pressure. It is considerably dearer, costing about 42s. a carat, while carbons may be had for 26s. a carat. About six to eight weeks' constant work suffices to wear out the setting of a drill. The working capacity is about 8 ft. a day in quartz and granite, to 10 ft. in sandstone and slate. Excellent drills may be had from C. Appleby & Co., 89, Cannon Street, and Calvert, Cornes, & Harris, 30, Walbrook, London.

Shafting.—It is quite unnecessary in a work of this character to enter fully into the question of sinking and timbering shafts, as these are operations which no man would undertake without being a qualified mining engineer. At the same time attention may be called to the accompanying drawings (Figs. 127, 128), which exhibit a special form of timbering wet shafts which has been adopted with great success in a

large Australian mine. The object sought to be accomplished was the damming back an immense quantity of water which poured into the



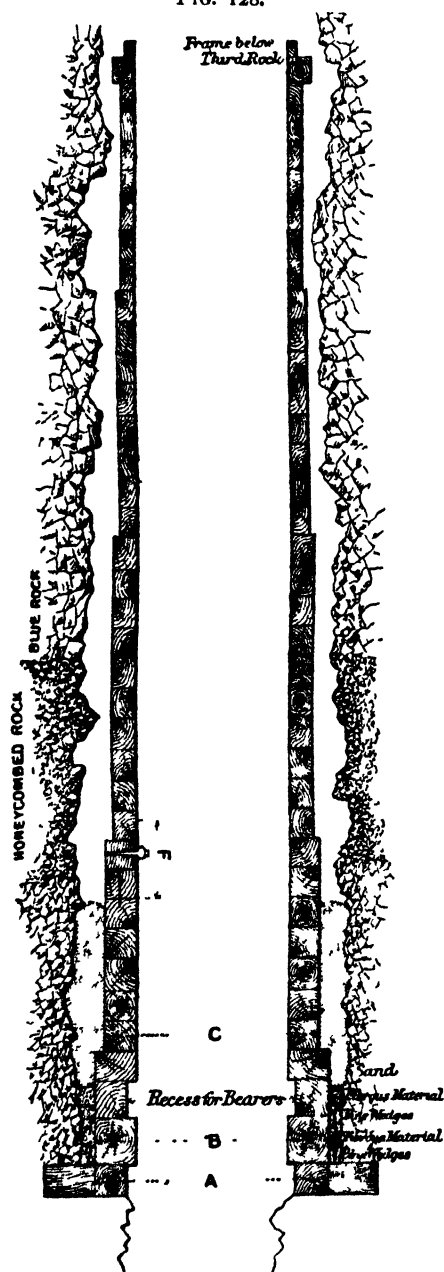
shaft from a honeycombed portion of the fourth layer of basaltic rock. The plan comprises a strong frame securely fixed in the bottom of the

shaft, and caulked round with fibrous material between the frame and the rock, so as to make a tight joint. The frame being secured, the superstructure is then built of strong timber bolted together and made water-tight, no clay being used.

Further information on tunnels and shafts will be found in André's 'Mining Machinery' and in Drinker's 'Tunnelling, Explosive Compounds, and Rock Drills.'

➤ *Drainage.*—According to C G. Yale, the principal methods employed for extracting water from mines in California are: Cornish pumps, baling tank, steam (or compressed air) pumps, and hydraulic pumps. For shallow mines, where there is little water, iron water buckets are used. The best form is barrel-shaped, for use in shafts or winzes, where there are no guides—this shape preventing catching on projecting timbers or rocks. The water bucket has a fixed bale; it does not turn over, but is emptied by lowering until a spindle in its bottom rests on the bottom of the water box, when the spindle opens the valve in the bucket bottom. This valve opens automatically when filling, and automatically closes when the bucket begins to rise. Oval and cylindrical baling buckets are also made for work in vertical shafts, and are arranged with shoes for wooden shaft guides. They are sometimes

FIG. 128.



TIMBERING WELL SHAFTS AT BALIARAI

arranged for rope guides also. Rectangular baling tanks are also made, arranged for self-dumping. These tanks are so constructed as to discharge themselves on arriving at the station, without reversing the engine, and there is no man required to let the water out. Such tanks are made for wooden shaft guides.

Water skips are of the same character for inclined shafts. The bale is attached to a pivot behind the centre of gravity, so that it will tip when it is run over the dumping track. The dumping is accomplished by making the rear wheels with a projecting face of smaller diameter, which catches on the dumping track and elevates the back end, and the skip is thus tipped to a proper angle for dumping. When lowering, the rear wheels must necessarily pass down first, returning the skip to its proper position. These and self-dumping skeets can be used for ore and water both, or for either. The self-dumping skeets are designed for use in vertical shafts. For sinking purposes they are very handy, for they can be shovelled into or baled into. In many cases, pumping from the bottom of a shaft while sinking can be dispensed with, as the water and dirt can be put into the skeet at the same time. These skeets are made with steel or iron boxes, and with single or double bar frames of Norway iron. They are self-dumping.

Tanks of this character will handle a great deal of water, and can be replaced by cages when not needed. They are too simple to get out of order often, and are readily repaired. In many places they are preferred to pumps, and are frequently kept in readiness as a substitute for Cornish pumps when the rods of the latter break.

Local practice varies greatly as to the means adopted for handling water. Some one form of pump does well in the first of the mines of the camp using one, and others follow the example set. Cornish pumps are more frequent in old mining districts than in newer ones, and in larger mines than in smaller ones.

For deep mine pumping, the Cornish bucket and plunger pumps are very extensively used. They are driven from the surface by a system of gearing directly connected with the engine shaft. The main gear is connected to the head of the bob by a pitman, one end of which engages into a wrist-pin set into one of several bosses cast on to the arms of the main gear, so placed as to give any desired throw to the pumps. Where the depth of the shaft is so great that one pump is insufficient to raise the water to the surface, a system of two or more pumps is made use of, the lowest one pumping into a tank station at a convenient height above it, from which the pump next overhead in its system takes its suction, and discharges either into a similar tank or directly out at the surface.

For depths below 300 or 400 ft., and for the handling of large volumes of water, the pumps generally preferred are those termed "Cornish

bucket" or "Jack-head," and the plunger pumps. In the jack-head pump the working bucket is packed with leather, which is secured to the body of the bucket by a brass ring. The suction chamber is bolted directly to the lower end of the working barrel, and has the suction pipe and wind-bore bolted to its lower flange. The "goose-neck" bolts on the top of the barrel, and has a bonnet and stuffing box through which the bucket rod passes. The upper clack sets on the branch of the goose-neck, with the column bolted on its upper flange. The clack chambers have doors, through which the clacks are removed for repairs. A door is sometimes made in the goose-neck, through which the working bucket may be taken out without taking off the bonnet from the top.

Jack-head pumps are best adapted for lifting water from cisterns or permanent sumps, although they are frequently used in sinking.

The common lift pump is different from the jack-head pump in that it has no goose-neck. The bucket-piece is bolted on the top of the working barrel, with a door for the removal of the bucket. On this, in a direct line, is set the column, with the pump-rods inside of it. The working bucket is the same as that for the jack-head pump. Of course there is only one clack. The advantage of these pumps in sinking is that the bucket can be lifted through the column for repairs while the whole pump is submerged in water.

The connecting rods through which the pump takes the motion are made of heavy timbers, strapped at the joints with iron plates and bolted through, and connect at the upper end with the pitman attached to the nose-piece of the bob. This pitman is furnished either with a crosshead or guide rollers, to keep it in position. The usual speed at which these pumps is run is about 12 strokes per minute.

The pumps employed in the deep shafts in California are all of the same general character, and do not differ materially in principle from those used in deep mines in other countries. They are either lift pumps or force pumps. In a complete set of deep pumps the two kinds are combined, the former being applied to raising the water from the bottom of the shaft to the height adapted to the capacity of a single pump; the latter forcing the water upward to the stations, or to the point of discharge.

It is unnecessary to go into any description of the numerous forms of engines which are employed to drive pumping gear. Different makers claim special advantageous features, and no one type is in exclusive use. The ordinary single-gearred pumping engine has the pinion placed on the engine shaft, gearing into the spurwheel, to which a pitman rod is attached. In the double-gearred engine two pinions are placed on an engine shaft and the pitman-rod pin is placed between the two spur wheels

thus adapting it for very heavy work. The simplest form of direct acting engine consists of a single high pressure cylinder connected directly to the main-bob; the king-post of this is larger than the nose piece, so as to give the engine leverage on the pumps. The valve gear is generally so arranged and adjusted that steam can be cut off at any portion of the stroke, or the stroke of the piston can be shortened. In some of the horizontal direct-acting compressed engines the pitman is connected directly to the crosshead and king-post, the connections between king-post and flywheel being made by two side rods.

Sometimes two lines of pump rods are used in the same shaft, working a double line of pumps from the same bob, in which case no counter-balance is necessary. Often, too, there is a double line of plunger pumps, when the main pump-rod extends down between the pumps to the next set below, and so on until the bottom pump is reached. An angle plate is attached to each side of the rod by bolts common to both, and to each of which the pump plungers are attached.

The ordinary surface bob is made principally of wood. The irons for the king-post and nose-piece are each made on two pieces, with sockets for angle-braces, and are securely bolted together by bolts running through the timber. The saddle-plate is made in three pieces, and admits of the trunnion shaft being placed on a line with the nose and tail-pins. The trunnion shaft is made of hammered iron, and is secured by running through the beam, and being keyed to the side and saddle-plates. The whole is drawn together by wrought-iron straps and turn-buckles. The back of the beam is extended to admit of a box being attached to be filled with old iron, rocks, or other heavy substances, for a counter-balance. These bobs are sometimes made wholly of iron. The balance bobs, made for use under ground, also vary in form.

In California, steam has been replaced by water power for running the pumping rig of several important mines.

A form of pumping apparatus recently introduced in California, is the Knight hydraulic pumping engine. It is especially designed for pumping water out of mines to any depth where there is water pressure above the drain tunnel. The hydraulic engine (or wheel) can be bolted to the shaft timbers at the drain tunnel, and a pump-rod run down to a special plunger pump made for use in connection with the Knight engine. By using this engine at the drain tunnel, the gears, bob, &c., below, are saved. Both the engine and pump can be set on the same bed-plate if desired, and the whole apparatus secured in the shaft below the surface in drain tunnel. After operating the engine, the water is exhausted into the discharge pipe of the pump. The only gain in this is the doing away with the pump-rod. Nothing is gained in power.

Steam force or plunger pumps are now much more largely used than

formerly, having been much improved and specially adapted for mining purposes. There are many forms more or less similar in general appearance and design, the different makers adopting details of construction originating from special experience arising in practice under different conditions of use.

For opening up mines, re-prospecting abandoned ones, working them under bond, temporarily, etc., the steam pumps are very advantageous. A steam pumping rig has the advantage of greatly reduced first cost over the Cornish system, and lately the makers of steam pumps claim equal economy with that plan. There is no doubt of one thing, that twice the number of mines have been opened lately, with steam pumps, than would have been the case had expensive Cornish rigs been absolutely necessary, the plant of steam pumps costing so much less originally. By their use a mine can be opened and tested. Moreover, there are many locations which it would not pay to test if steam pumps could not be obtained to do it with.

Horizontal plunger pumps are mainly used for station pumps, and are made to raise the water on a single lift as high as 1000 ft. The particular advantages of the direct acting system are the first cost, ready means of placing in position, &c., especially for prospecting purposes. The steam is brought down the shaft in jacketed pipes, and the pipe column is brought up to the surface, the pump being below. The best makers now construct these pumps so that the parts are readily accessible, and muddy water can be raised.

A class of vertical sinking steam pumps has come into use quite largely of late, having been much improved. Their peculiar feature is the special adaptability for shaft sinking purposes. They are of comparatively light weight, portable, and require a very limited space for operation. A steady continuous stream of water is thrown, and there is very little jar. They operate in any position, whether suspended from the ringbolt attached to the steam cylinder head, hanging on timbers by means of the adjustable dogs founded for the purpose, or placed horizontally. The plunger barrel is easily removed for substitution. A peculiarity of this pump is the suction condensers, the utility of which is an important consideration in sinking a shaft, as it not only consumes exhaust steam, avoiding the use of exhaust pipes leading out of the mine, or the damaging effect on the timbering if allowed to escape into the shaft, but is so constructed as to increase the efficiency of the pump by doing more work with a given amount of steam.

A recent application to vertical pumps of this class, by James E. Dow, of San Francisco, is a governor, so that if the water is lowered so that air gets in, the pump cannot "run away" and be damaged. This new feature is important. With speed governor and suction condenser, this

type of pumps is greatly improved. None of these attachments is complicated, and the pumps may be managed by any ordinary mechanic.

In the very deepest mines on the Pacific Coast, such as those on the Comstock, Nevada, it was found that the weight of the pit work became so enormous that finally the rate of speed, which is determined by the weight of the mass set in motion, had to be decreased beyond all practicable working limits. The question then presented itself: What system of pumping can be adopted? The hydraulic system was suggested, and has proved successful in the Comstock and several other mines.

The general plan of this system will be rendered plain by reference to the accompanying drawings, Figs. 129, 130, 131.

The mine to which this system is applied is 2400 ft. deep; the water to be pumped is to be raised from the 2400 level to a height of 800 ft., and discharged into the Sutro Tunnel, through which it is run off. Upon the 2400 level is erected a pair of hydraulic pumping engines *a*, which receive their pressure water through supply pipes from the surface. By these two pumps the drain water is raised through discharge column to the Sutro Tunnel, and the water used in doing the work of pumping is sent back through the return pipe to a reservoir *b* on the surface. Upon the surface there is a cast-iron accumulator *c*, which is 60 ft. high and 25 in. in diameter. In this accumulator there are but 20 ft. of water, the remaining space being occupied by air.

This air is kept constantly at a certain pressure by means of plunger pumps *d*, which in turn are operated by a compound steam engine *e*. These pumps, which supply the accumulator with water and keep it at a pressure great enough to run the two underground pumps *a*, take their suction from the reservoir *b*, into which the return water is discharged.

Now, since this pipe which supplies the pumps *a* is of such great length, and moreover, since the whole mass within it must be put in motion and brought to rest again during each stroke of the pumps *a*, it is evident that some mechanical contrivance must be introduced for the purpose of lessening the jar in the pipes caused by the sudden stoppage of such an immense mass of water. Let *e* be the valve which shuts off supply water from *a*. Close to the underground pumps there are firmly fixed two cast-iron cylinders *f*, in which the plungers *g h*, carrying the crosshead *i*, are free to move. On either side of the arm *i*, are strong springs *j*, held in position by the abutments *k*. Now, when water in supply and return pipes is at rest, the arm *i* is exactly in the middle between the cylinders *f* (since the plungers *g h* are so proportioned that the total pressure on *g* is precisely equal to total pressure on plunger *h*). But after the underground pump *a* has completed its stroke, and the total mass of water in the supply pipe has to be brought from a velocity = τ

FIG. 129.

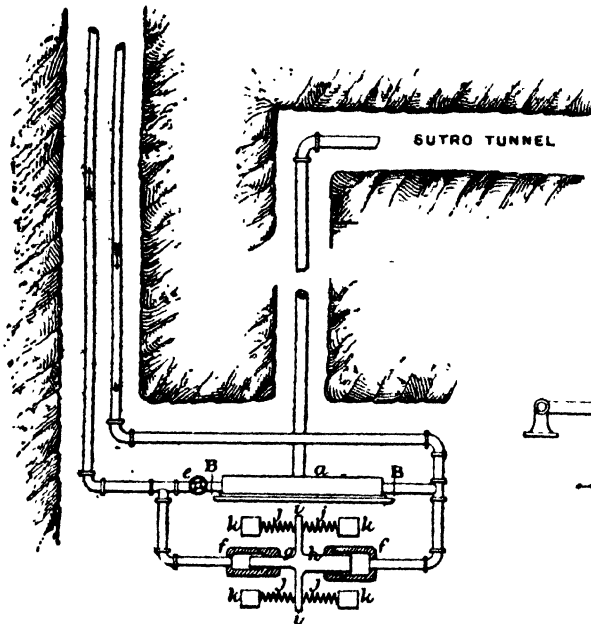
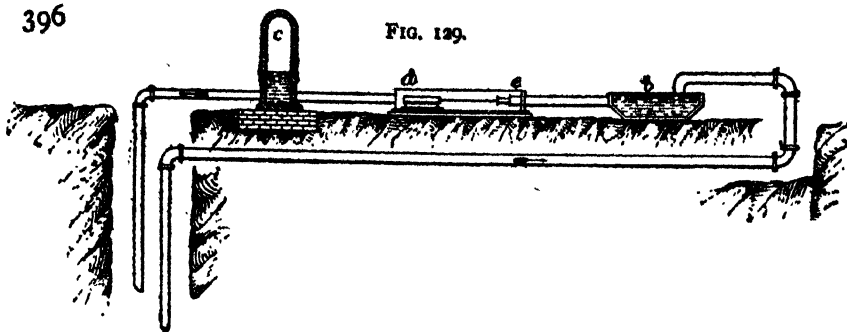


FIG. 130.

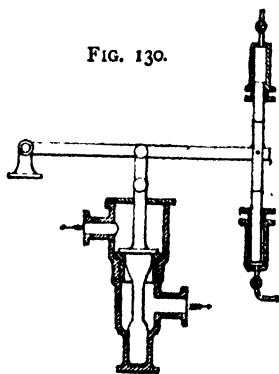
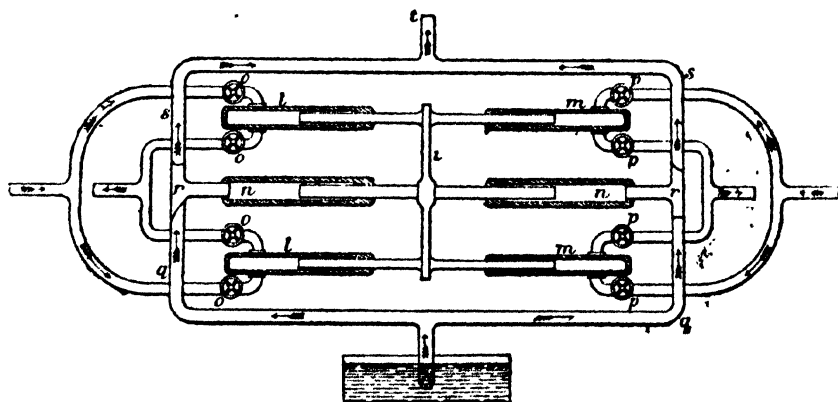


FIG. 131.



HYDRAULIC SYSTEM OF DRAINING MINES.

to a velocity equal to zero, the inertia of this mass (due to velocity v) is supposed to be gradually reduced to zero by the plunger h being forced (on account of extra pressure on g) over to the right side. The distance it travels to the right depends, of course, upon the extra pressure produced by checking the velocity of supply water, and also upon the strength of the springs j , which begin to be compressed as soon as g h begin to move. Now, after the crosshead has been forced over by this extra pressure to its maximum extent (the water in the three columns also coming to rest at this moment), the compressed springs j will immediately react to place the crosshead and plungers in their central position, ready to again take up the inertia of the mass in supply column on the return stroke of the pumps.

Thus, the object of these springs j is not only to assist the water in the return column in taking up the inertia of the water in the supply column, but chiefly to replace the plungers g h in their central position after the masses in the three columns have come to rest.

The general arrangement of the system having been described, an account will now be given of the manner in which the work is performed by the pumps a . In Fig. 131 is shown in detail that portion of the underground pumps lying between the lines, B and B, on Fig. 129, leaving out, of course, that portion already explained for taking up the inertia of the supply water.

There are two of these underground pumps, situated on the 2400 level of the mine, but since they are similar to and work independently of each other, it is necessary to describe but one of them. They both take their supply water through the valve e . Acting independently, as they do, they will not necessarily commence and finish their strokes at the same time. But, in order to discuss the system under its extreme conditions, consider the two pumps as working together, thereby making the velocity of the supply water and the inertia of the moving masses a maximum.

Therefore, in each pump, let l and m be the four pressure cylinders, and n the pump cylinders for raising the drain water to the Sutro Tunnel. Let o be the valves leading to the pressure cylinders from the supply and return pipes, and let the plungers be connected with a crosshead (as shown in drawing), so that when any movement of the plunger m takes place, the plunger n will be carried along also.

Now to follow the operation of these pumps, first assuming that the pressure of the water in the supply pipe is great enough to produce motion of the plungers l .

First, suppose the valves o to be opened (valves p remaining closed). The two plungers l will be forced over to the right, carrying with them the larger plunger n . While n travels to the right, the space left by it

is immediately filled with drain water coming through the suction pipe *r*. Now, suppose valves *o* be closed and the stroke finished, the valves *p* are then opened, and the pressure of the supply water against the plungers *m* forces them to the left, and the water remaining in cylinders is forced through the valves into the return pipe, and thence into reservoir *b* upon the surface. At the same time, the water which was drawn into cylinder *n* on preceding stroke, is now forced (on account of the check valve *r* closing) through *s* into discharge column *t*, and thence 800 ft. into the Sutro Tunnel. Again, while this stroke to the left is being made, the space left by plunger is being filled with drain water through suction pipe *r*. At the completion of this stroke, valves *p* are closed and valves *o* are opened, the crosshead and plungers again move to the right, and the water remaining in the cylinders is again forced through return column into reservoir. At the same time, the water in cylinder *n* is forced upward (on account of check valve *q* being closed) into *t*, and thence to Sutro Tunnel. Thus the operation continues, the valves being worked automatically by means of tappets carried by the crosshead. The number of strokes made by these pumps evidently depends upon the pressure of water in the supply pipe, and also upon the rapidity with which the valves are opened and closed.

It will be seen that the sole object of these two underground pumps is to raise the drain water from the 2400 level of the mine to the Sutro Tunnel, a distance of 800 ft. The object of the accumulator *c* upon the surface, is to accumulate the water within it, under a pressure great enough to work the pumps *a*, and the object of the compound steam engine and pumps *d e* on the surface, is to supply the accumulator with exactly the same amount of water taken from it, to run the underground pumps. Furthermore, it will be noticed that the same amount of water which is used to run these pumps *a* is returned to the reservoir on the surface, from whence it is forced into the accumulator by pumps *d* to be used over and over again.

Here it might be well to mention the fact, that the surface pumps *d* were constructed with a capacity great enough for two more pumping engines, similar to those just described (*a*), so that when a depth of 1000 ft. more was reached, this extra set of pumps would be put in at the 3400 level, and the process go on as before.

The above description explains the action of the hydraulic pumps on the Comstock. Since it was written, the pumps have been running several years, and the springs *j* have been replaced by air chambers, so that the concussion of water lost in discharge, pressure, and return columns, has been overcome by means of air chambers erected on the same station with the pumps. It was found impossible to properly

regulate the springs to work as efficiently as the air chambers have done.

Davey remarks (Jl. Soc. Arts), as a curious fact, that although the plunger pump has been almost universal in Cornwall for the last 50 years, it has, until lately, been little adopted in collieries. It is usually employed to force the water 40 fathoms only, so that in a shaft 200 fathoms deep there would be five pumps, lifting the water from the bottom in stages, from one to the other. The lowest pump is usually a bucket pump, for convenience of sinking. Stage pumping is often convenient where feeders of water occur in the shaft at different levels, because, as a question of power, it is more economical to pump the water from where it is found, instead of letting it run to the bottom of the mine, and pumping it from there ; but stage pumping is generally very troublesome, and requires very careful supervision. It becomes necessary to have a pipe from one pump to the next pump below it, so that the water may be run down to keep all the pumps equally charged under the varying conditions of wear and tear. Trained men are kept for this and the other work connected with the pumps. In Cornwall they are called "pitmen," and in the north of England "pump-doctors."

The plunger pump has, of late years, been so improved in form, that it may be safely employed to pump water even as high as 200 fathoms in one direct lift, so that, as far as the pumps are concerned, the necessity does not remain for stage pumping. The old form of plunger pump was not suitable for pumping to a great height in one lift. It was weak in form, and inconvenient of access to the valves, and involved risk of life to the pitmen, because all the operations of changing valves and repairs to the pump had to be done in the shaft. The modern and most approved method of draining mines economically, without risk of flooding, is to have two pumping engines, each capable of draining the mine, and provided with plunger pumps.

In the Cornish engine the practical limit of expansive working is soon reached. Not so in the compound engine ; and as the compound engine is double-acting, and the Cornish engine necessarily single-acting (when worked expansively), the compound has distinct advantages, both as regards power and economy. Of late years the compound engine has met with a large application in mines.

As a recent example of heavy pumping performed by a compound engine, with the improved plunger pumps before described, Davey selects one of the engines recently erected by the South Staffordshire Mines Drainage Commissioners. This engine will raise 4,000,000 gal. per day from a depth of 600 ft. It has cylinders 52 in. and 90 in. diameter, by 10 ft. stroke, and works 27-in. diameter plunger pumps, 10 ft. stroke, equal in power to a 120-in. Cornish engine.

It is not unusual to find in collieries as much as six tons of water to one ton of coal—that is to say, there are collieries raising say 6,000 tons of coal per week and 36,000 tons of water in the same time.

The conditions of mining vary so much that it is impossible to frame any general rules as to the type of machine which is best suited to any particular mining operation. The old form of pumps, worked from spear rods, gave considerable trouble, and involved costly maintenance; considerations which favoured the introduction of underground pumping engines.

In underground pumping engines, the engine and pump are combined in one self-contained machine, employed to force the water from the position in which it is placed in the workings, direct to the surface through a single pipe. Sometimes the boilers for working the engine have been placed in the workings near the engine, but generally steam has been taken down through a pipe from boilers situated on the surface. It is seldom that very heavy pumping is performed on this plan. In many situations it is convenient and desirable, but from an economical point of view there is a serious loss by condensation in conveying steam down the pit. Wherever this plan is employed, care should be taken to dry the steam as much as possible before it enters the cylinders of the engine.

Davey emphasises the fact that there is no greater impediment to steam-engine economy than the presence of water in the steam, or what is practically called “wet steam,” arising either from priming of the boilers or from subsequent condensation in the steam pipes.

Steam is often conveyed down the mine for the purpose of working not only the pumping machinery, but also the hauling engines; but since the introduction of compressed air machinery, the practice is not so usual.

Some years ago Davey erected, at the Clay Cross Collieries, a pair of underground engines to force water 1000 ft. in one direct lift, and experiments were then made to determine the loss by condensation in taking steam down the pit, with the following results:—

Diameter of pipes	7½ in.
Vertical height of pipes in pit	915 ft.
Horizontal length on the surface	185 „
Total length	1100 „
Length of pipes with non-conducting cement	1015 „
Length of pipes unclothed	85 „
Pressure of steam at surface	45 lb. per sq. in.
Pressure of steam at pit bottom when engines were standing	46 „ „

The engines consisted of a pair of single-cylinder direct-acting engines.

Diameter of cylinders	20 in.
Length of stroke	36 "
Speed of engines	10 double strokes per minute.
Water from condensation in the pipes	12 cub. ft. per hour.
Water from condensation when the engines were standing	8 " "

The steam pipes stood on a strong piece of timber at the pit bottom, and were kept in a vertical position by stays fixed 27 ft. apart vertically. An expansion joint was provided near the top, and the expansion of the pipes was found to be 18 in., or 2 in. for every 100 ft. in length.

The underground workings of mines extend to great distances, and cover large areas, necessitating a very large application of power. Compressed air forms a very convenient medium for actuating underground machinery, and is extensively used for working hauling engines. Mechanically speaking, it is not economical; but as long as it is commercially profitable, there is a great incentive to the engineer to increase, if possible, the efficiency of its application. The general practical efficiency is from 25 to 33 per cent. of the power applied. The following results of experiments made with a pair of air compressors working underground hauling-engines, are of interest, and highly instructive, and fully demonstrate the increasing loss of efficiency with the increasing pressure to which the air is compressed: When the air was compressed to 54 lb. per sq. in., the total efficiency amounted to only 14·27 per cent., but when the pressure of compression was reduced to 40 lb. per sq. in., the total efficiency rose to 24·42 per cent. The experiments were conducted under precisely similar conditions, but the low efficiency in both cases is accounted for by the fact that the hauling-engine did not use the air expansively. The utilised power in the steam-engine, as ascertained from the steam and air indicator diagrams, was 87·25 per cent. in the first case, and 85·62 per cent. in the second. As all other things were equal, the lower efficiency resulted from the increased heat formed by the higher compression. The loss of power arose from the cooling of the air after compression, before reaching the hauling engines, and the want of expansion of air in the hauling cylinders. It has often been pointed out that the loss from cooling might be partially restored by heating the air before it enters the hauling cylinders, but no practical application of the idea is before the public.

Hydraulic power has been used to a considerable extent for the transmission of power in mines, especially for raising water from the lower workings to the pumps of the main pumping engine, and in some cases actually pumping the main water feeders of the mine. It has also been used for working hauling engines. An interesting example of such an application has been recently instituted at Marseilles, in a colliery. The engineer of the mine has, since the completion of the installation,

carried out careful experiments, the results of which yield the following facts. The installation is specially interesting, as it was designed to meet special difficulties. The portion of the workings to be drained was subject to periodical inundations. It was therefore out of the question to think of employing underground steam-engines. It was also impossible to employ pumps in the shaft, already fully occupied with pump rods and winding cages. To meet the special difficulties of the case, Davey proposed hydraulic transmission from the surface. In a gallery which is subject to inundations, are placed two hydraulic pumping engines, delivering the water which they pump into the drainage gallery of the mine some distance below the surface. At the surface of the mine is placed a compound condensing pumping engine, actuating two double-acting pumps, from which water is conveyed at a pressure of 42 atmospheres direct to the hydraulic pumps. The water supplying the hydraulic pumps is not discharged into the mine, but returns again through a second pipe to the pumping engine (or heart of the system) on the surface; so that the water which actuates the hydraulic engines simply circulates through the system by means of a supply and return pipe, just as blood in the veins of a human being returns to the heart to be pumped, over and over again.

It is evident that there is a back pressure on the hydraulic engines, due to the vertical head of water in the return pipe. The effective pressure is the pressure produced by the pumping engine on the surface.

In Sir William Armstrong's system of hydraulic transmission, the pumping engine delivers its water into an accumulator, from which power is distributed. The accumulator is here introduced to store the power of the prime mover, and to meet the interrupted demand for power caused by the intermittent use of the hydraulic machinery. In the case under notice, the use of the hydraulic pumps is constant; there was, therefore, no object in storing power. The main desideratum was to cause the hydraulic pumps to respond directly to the motion of the engine on the surface, so that the speed of the engine on the surface should give the speed at which the hydraulic engines were working underground. It will easily be seen that if the pumping engine on the surface delivered a constant stream, and the hydraulic engines below used a constant stream, there would be no necessity for anything more between the two engines than a closed pipe; but both engines have a reciprocating action, and, therefore, produce an intermittent stream in the pipe. To correct this intermittent action, and to preserve a constant pressure, and to maintain the engines working in unison, Davey devised a steam regulator. The regulator simply consists of a plunger in direct communication with a pipe connected to a steam piston, on the face of which is maintained a constant steam pressure. The pressure of the steam on the piston, multi-

plied into its area, being equal to the reacting pressure on the plunger multiplied into this area, there is, therefore, a balance of forces. Any rise in pressure in the supply pipe, owing to intermittent action of either of the engines, causes the plunger to move downwards, and therefore increase the capacity of the pipe, and *vice versa*, whenever reduction of pressure occurs. The movement, then, of the steam regulator, is to preserve a constant pressure by varying the capacity of the pipe. It also serves the double purpose of a governor on the steam-engine. The throttle valve of the steam-engine is connected directly to the piston of the steam regulator. A little thought will enable one to see that a dead weight on the plunger, instead of steam pressure, would not have the same effect. The inertia of the weight would cause heavy shocks in the pipes. The weight of steam is so much less than its pressure that its inertia may be neglected.

The total efficiency of the whole system reaches 54·2 per cent., and the consumption of steam amounts to 13·76 kilogrammes per horse power in water pumped. Davey predicted a consumption of 13·60 kilogrammes. The total horse-power exerted by the steam-engine was 229·2 horse-power, and the work actually done by the hydraulic engines 124·2 horse-power.

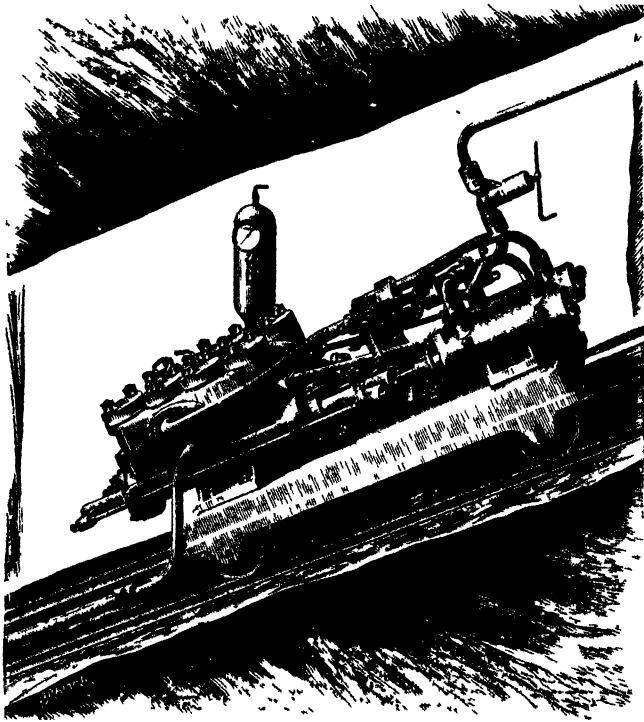
Among the very best known machines for raising water from narrow shafts is the Deane pump, made by the Pulsometer Engineering Company, Nine Elms Iron Works, London. It is simple, efficient, and quiet in operation. The pump-bucket is placed down the shaft near the water level, whilst a plunger may be put at the surface, thus making the engine deliver at both up and down strokes. The Deane sinking-pump is easy and noiseless in operation, and has no water-piston to wear leaky. It is provided with dogs by which it can be suspended from the mine shaft timbers, and is made in small parts for easy transport on mule-back. The Deane double-plunger sinking-pump requires no platform or foundations, and has both stuffing-boxes at the upper ends of their respective plungers, thereby allowing all the grit to fall away from instead of towards the packing. The valves are easily accessible. The water passages are very large and almost in a direct line. These pumps are adapted to lifts of 150 ft. and upwards. The Deane plunger-pumps for mining have two plungers working in opposite ends of a water cylinder divided in the centre. They are adapted to situations where the heavy pressure or gritty nature of the water precludes the use of piston-pumps.

The pulsometer made by this company is sufficiently familiar to miners. It will work just as well suspended from a chain or rope, as if it were permanently fixed; its very small size, in proportion to the quantity of water thrown, making it available in a great variety of situations where no other pump can possibly be applied, giving satisfaction in

the most awkward positions and for the most trying work. Should a sudden influx of water occur in any sinking operation, and the pulsometer be drowned out, it can be at once drawn up to the top and set to work again without delay. It is far easier to hang up a pulsometer by a chain down a quarry side or in a well, with one length of rising main attached, and a flexible steam hose, than to sling any of the forms of direct-acting pumps on its platform, especially as in the latter case the exhaust steam has to be provided for, whilst the pulsometer disposes of its own. The shaft is, therefore, kept cool, and does not inconvenience the workmen. For this reason especially it is invaluable for underground pumping in mines, &c. The pulsometer will, where the conditions remain constant, work day and night without attendance—in fact, as long as steam and water are supplied to it.

Fig. 132 illustrates one of a pair of Fielding's duplex pumps, with

FIG. 132



FIELDING'S DUPLEX PUMP

hydraulic motor. These pumps had quadruple rams $3\frac{1}{2}$ in. diameter by 6 in. stroke, the hydraulic cylinders being 3 in. diameter. The working

pressure was 1500 lb. per sq. in., and this was given by the surface pumping engine, which had two steam cylinders of 12 in. diameter by 12 in. stroke, with steam at 100 lb. initial pressure, the surface pump easily supplying these two pumps. The underground pumps were each equal to raising 60 gal. per minute, from an inclined shaft 330 ft. deep, and this work they did with great ease, and at small expense and loss. The pumps were supplied by Calvert, Cornes, & Harris, 30 Walbrook, London.

Fig. 133 shows the manner in which pumps of the "Cornish" type are used, both in vertical shafts and in slant workings. At the left hand side of the shaft are seen two pumps at different levels, both driven by the same T bob, the spear rods of the lower one being attached to those of the upper one. The pump in the slant working is driven in the same manner, with the intervention of a bell-crank arrangement. The general arrangement of this drawing was taken from pumps made by Hayward Tyler & Co., for the Tapada mine.

The pump body, with its valve boxes and air-vessel, is shown on larger scale in Fig. 134. The rising main pipes in the shaft are omitted to avoid complicating the illustration.

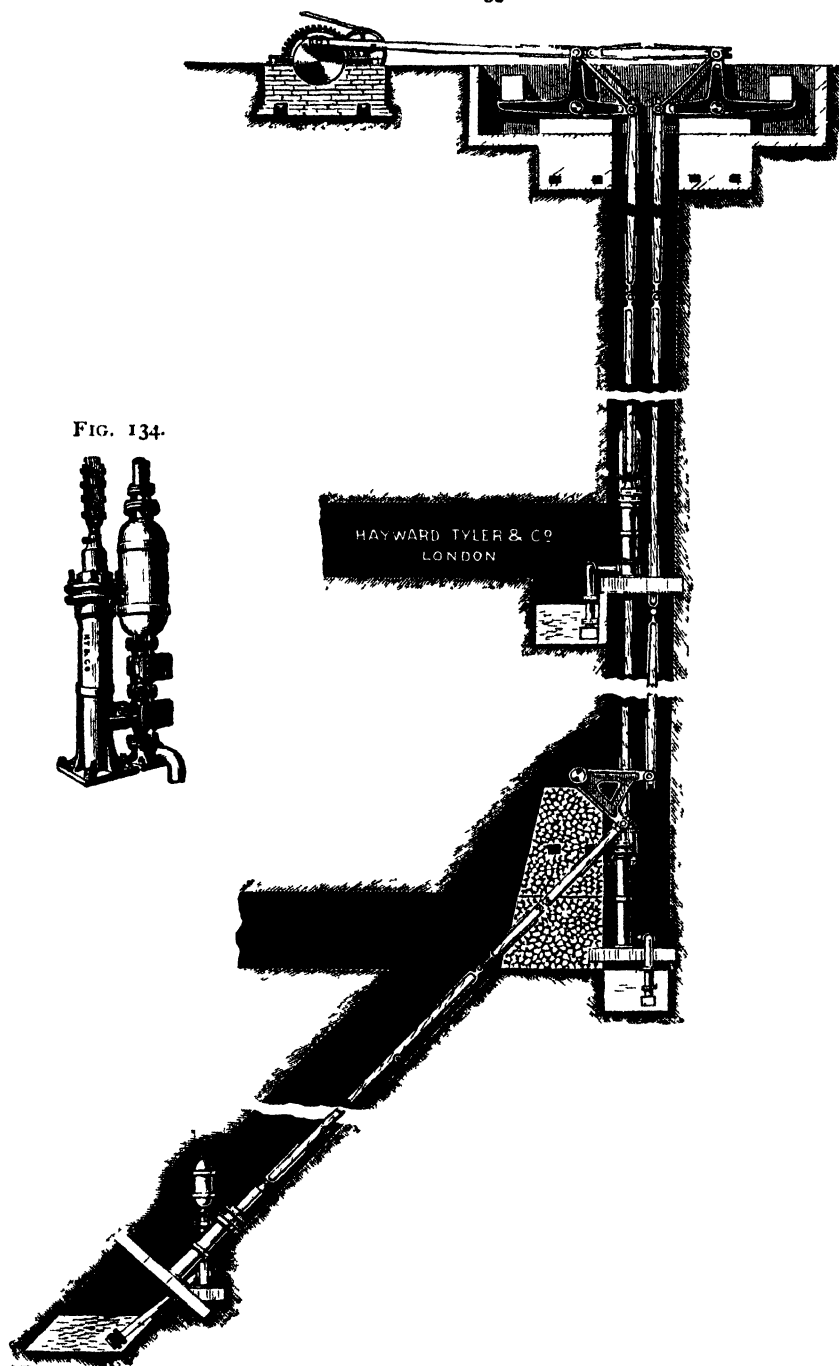
The T bobs which operate the pumps are worked by a horizontal connecting rod carried either direct from the engine or from a pumping frame driven by a strap as in Fig. 133. These connecting rods are sometimes laid for a long distance supported on rollers.

Fig. 135 illustrates an arrangement adapted for pumping where the water level is liable to great variations. The pump barrels containing the suction valves, and the buckets, are placed at the lowest level, and above them are fixed cast-iron pipes of somewhat larger diameter than the barrels, carried to a height above the highest probable water level, the pump covers and stuffing boxes are on the top of these stand pipes, so that the pump buckets can be drawn out for clearing, even if the water has risen many feet above the barrels themselves. This arrangement is occasionally of great value.

For general pumping purposes in mining work the direct acting steam pump has now come into almost universal use; the great convenience and compactness of this form of pump makes it very applicable, and many hundreds, or even thousands, are at work in mines. The pump is often placed a long distance from the boiler, the steam or compressed air being conveyed with comparatively little loss. Fig. 136 shows pump and boiler on same bed-plate suitable for water supply where the water has to be raised from a valley to the mines above.

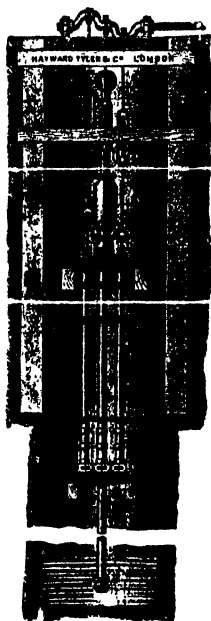
Another arrangement of great use where water is abundant in the valley and a supply is required above is to utilise the fall of the water in driving a water wheel or turbine which works a pump as shown in

FIG. 133.



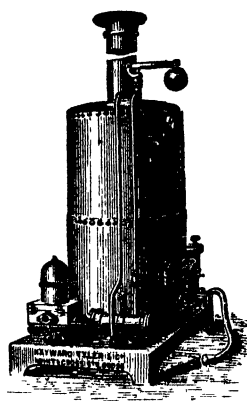
CORNISH PUMP WORKING UPRIGHT AND SLANTING.

FIG. 135.



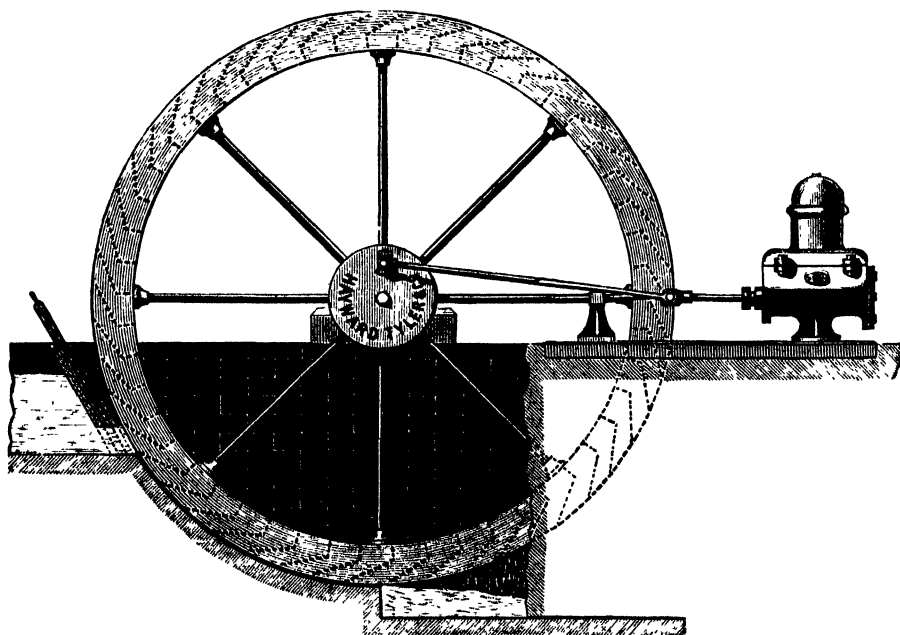
PUMPING WATER
OF VARIABLE LEVEL.

FIG. 136.



STEAM PUMP.

FIG. 137.



WATER-WHEEL PUMP.

Fig. 137. These water wheels are sometimes of great size, but considerable use can be made even of a small one.

Ventilation.—In metalliferous mines there are generally several shafts; the shaft in which the pumps are fixed is usually called the engine shaft. When there are two or more shafts communicating with the workings, it is a curious fact that a very considerable amount of natural ventilation takes place without any machinery or appliances whatever. If we take a lamp chimney and put in it a vertical division extending some distance down from the top, dividing the cross section of the chimney into two parts, we shall find that instead of having, as before, an upward current occupying the whole of the chimney, we shall get an upward current on one side of the division, and a downward current on the other. This action obtains in mines. If the shafts are the same depth, and come to the surface at the same altitude, it is difficult to say which will have the upward current. That will generally depend on an accidental circumstance. It is easy to see the reason of the current when we consider that the bottom of the mine is warmer than the top, so that when a current is set up in one shaft, that shaft becomes occupied by a rarefied column of air, whilst the other is filled with cold air of the normal density. The cold column being heavier than the warm column, a current is at once set up, the cold air entering the mine through one shaft, and the warm air leaving it through the other.

The method employed in collieries, till within the last few years, was generally a furnace placed at the bottom of an upcast shaft. That system was very wasteful as far as the consumption of coal was concerned, and also had other serious practical objections. Collieries are now generally ventilated by means of fans put in communication with the up-cast shafts, and means are employed to utilise the shaft for other purposes, sometimes for pumping, and sometimes for both pumping and winding. It is easily seen that it is necessary, in order that the fan may perform its functions, that the top of the shaft should be closed. When the shaft is used for pumping purposes alone, that is easily done by covering the shaft and working the pump rods through holes in the covering. When used for winding, an ingenious arrangement of trap doors is provided, so that as the winding cage comes to the surface it pushes open the doors, and closes them again on its descent.

The efficiency of ventilating fans, compared with furnace ventilation, is very great. The general total efficiency of mechanical ventilation by means of fans is from 40 to 50 per cent. of the power applied, and the consumption of coal per horse-power of effective work done should not exceed 8 lb. The consumption of coal for an equivalent work with the old furnace amounted to from 30 lb. to 60 lb. As an example of mechanical ventilation, a Guibal fan, 46 ft. in diameter, and 14 ft. 10 in.

wide, extracts from the mine 246,000 cub. ft. of air per minute. In the Guibal fan, the high velocity of the periphery of the blades is obtained with a slow rate of rotation by making it of large diameter ; but smaller fans are made to give a large displacement by increased rapidity of rotation. A Schele fan, 9 ft. 6 in. in diameter and 3 ft. 2 in. wide, gives a displacement of 106,000 cub. ft. of air per minute.

Motive Power.—When water power is available, the question of its best application is one of great importance and interest. A few general remarks on this subject will not therefore be out of place.

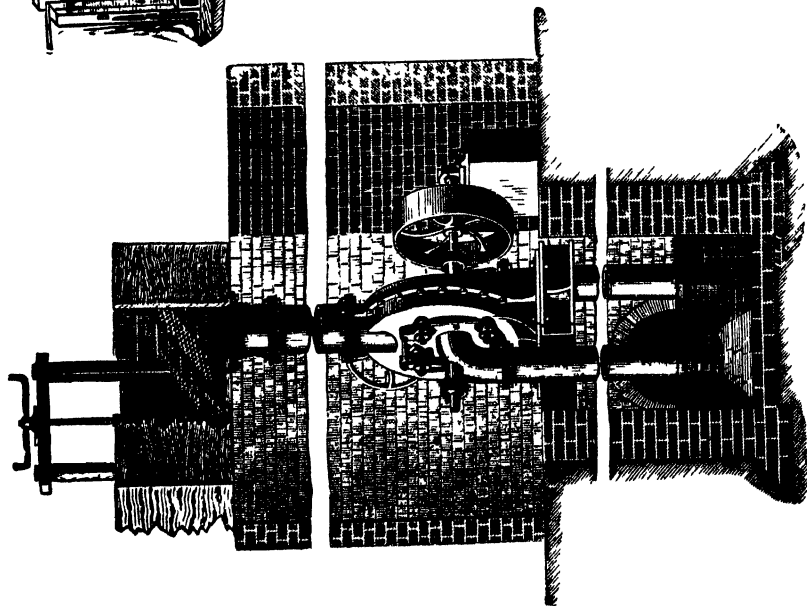
There are still some instances in which the old-fashioned water-wheel, so universally in favour in the days of our great-grandfathers, may be applied without serious disadvantage ; but, as a rule, it is too unwieldy and too slow. Its best efficiency does not exceed 66 per cent. ; much power is lost in the train of wheels necessary to bring the speed of the shafting up to what is wanted. Well designed turbines will give an efficiency of over 75 per cent., and are available for any fall, from 3 ft. up to many hundreds of feet. They should be made with adjustable orifices, so that the consumption of water can be economised to the utmost, and the power of the turbine adjusted to the varying necessity of the work. Some popular mining turbines work simply by the impact of a jet of water upon cups placed round the periphery of the wheel. The percentage of efficiency of such is low, and they should not be used except in cases where there is a very high fall and abundance of water. There are many classes of pressure turbines, the best being those in which the water is confined, under the pressure due to the head, in a case round the wheel, and from which it flows upon the wheel, and driving it round, passes through it, and escapes near the centre.

The vortex turbine (a favourite in many goldfields), invented by Professor James Thomson, is without question the father of the varied races of centre-vent turbines which have been brought into use. Many of these are cheap cast-iron copies, and in almost all the imitations, cheapness in construction has reduced the efficiency.

The vortex turbine (still manufactured by Gilbert Gilkes & Co., late Williamson Bros., Kendal, who were the sole licensees under Professor Thomson's patent) is divided by a centre plate, so that half the water escapes on each side, and exact equilibrium is maintained. This is of very great importance when the fall is sufficiently high to allow of the turbines being placed in the position shown in Fig. 138, the wheel being vertical and the shaft horizontal. Some other turbines are placed in the same position, but are liable to a heavy end thrust, as the water escapes only on one side of the wheel.

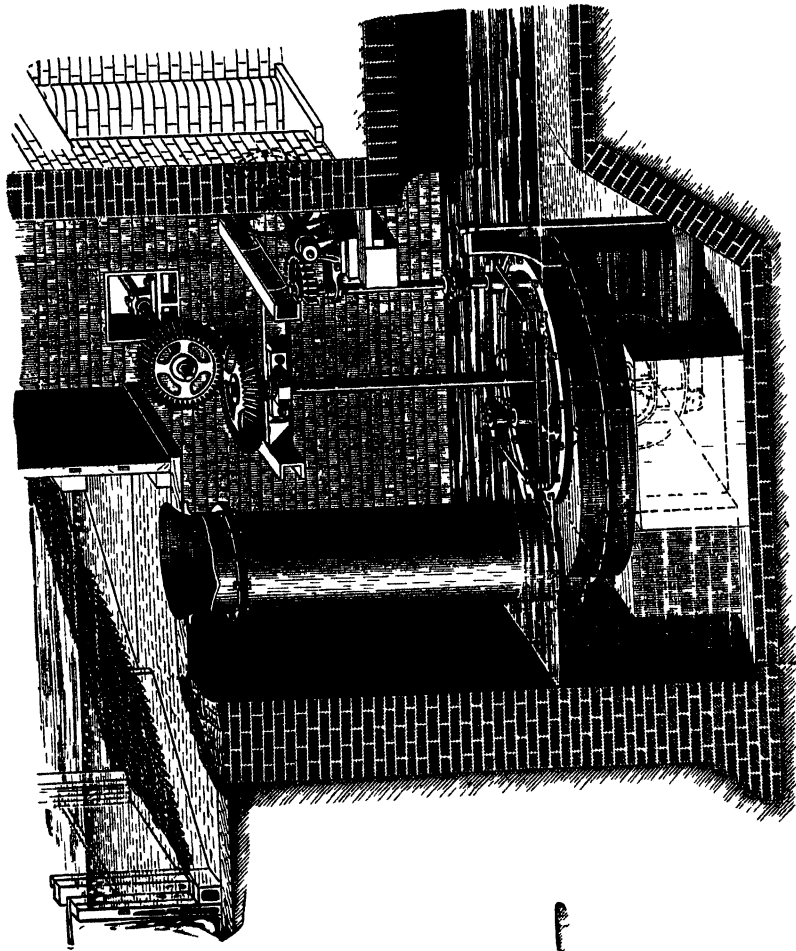
In Fig. 139 the turbine is shown in a horizontal position, at the bottom of the fall. This is doubtless the best position for low falls ; but when

FIG. 138



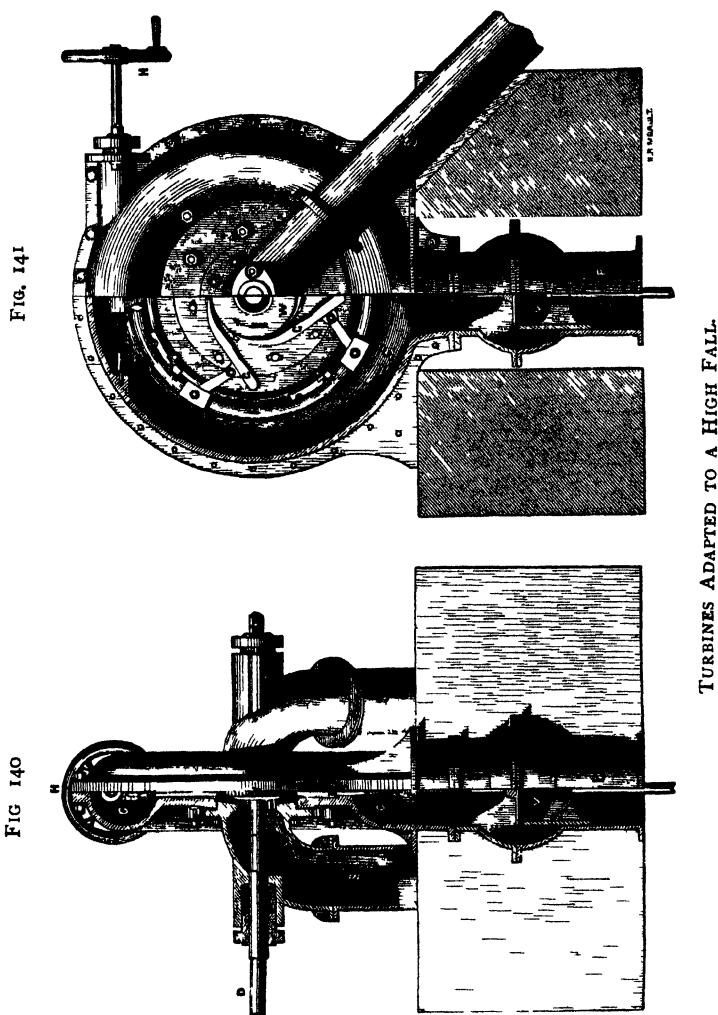
VORTEX TURBINE.

FIG. 139



TURBINE WORKING HORIZONTALLY.

the fall is high, it is a great convenience to place the turbine, as in Fig. 138, 15 to 25 ft. above the bottom of the fall, the height of the fall below the wheel being utilised by suction. The sectional drawings, Figs 140 and 141 will illustrate the construction of turbines of a high fall, arranged on this principle.



The velocity of a jet of water depends upon the height of the fall. When the fall is many hundreds of feet in height, the velocity is excessive, and unless the wheel be large, its speed will be excessive also. In such cases, the well-known Girard turbine should be used. It is fairly efficient, and can be regulated with a good deal of accuracy. Its

general construction is illustrated in Fig. 142. The diameter of the wheel can be varied to any extent to suit the requirements as to speed.

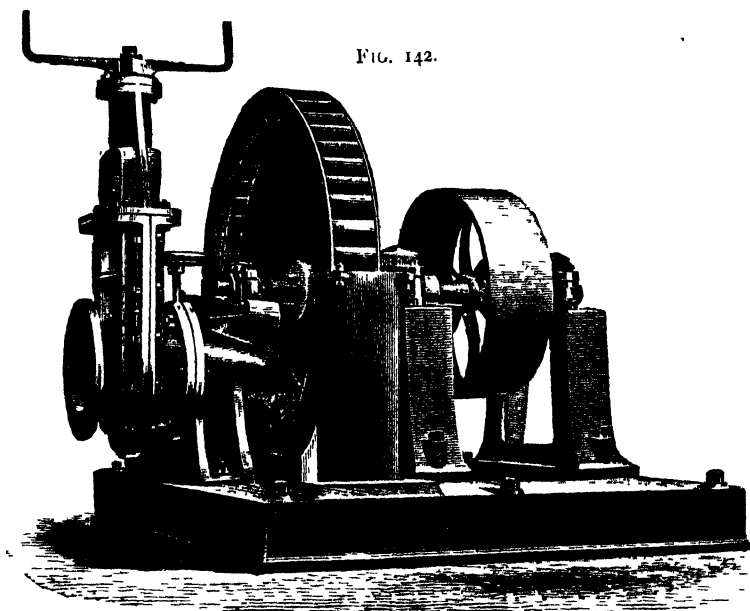


FIG. 142.

GIRARD TURBINE.

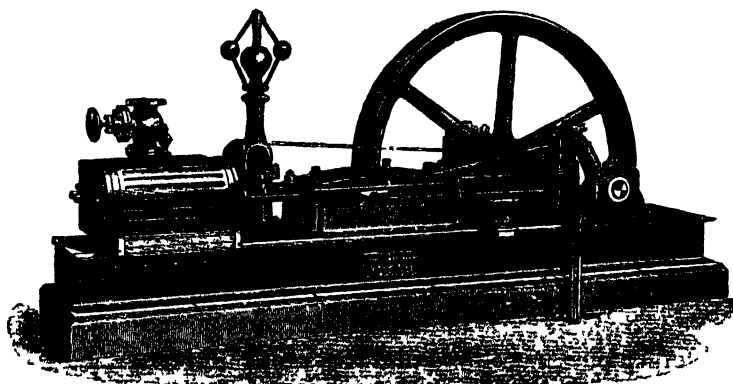
The following table states the quantity of water required (in cub. ft. per minute) for various powers under various heads.

			HORSE POWER.									
			8	10	15	20	30	40	50	60	70	80
3 ft. fall	..	..	1883	2353	3530							
4 "	"	"	1412	1765	2648	3530						
6 "	"	"	941	1176	1765	2353	3530					
8 "	"	"	706	883	1324	1765	2648	3530				
10 "	"	"	565	706	1059	1412	2118	2824	3530			
12 "	"	"	471	588	883	1176	1765	2353	2940	3530		
15 "	"	"	377	471	706	942	1412	1884	2353	2824	3295	
20 "	"	"	282	353	530	706	1059	1412	1765	2118	2471	2824
25 "	"	"	226	282	424	565	847	1130	1412	1694	1977	2260
30 "	"	"	189	236	353	471	706	942	1176	1412	1648	1883
35 "	"	"	161	202	303	403	606	806	1010	1212	1412	1612
40 "	"	"	141	176	265	353	530	706	883	1059	1235	1412
45 "	"	"	125	157	235	314	471	628	784	941	1098	1255
50 "	"	"	113	141	212	282	423	565	706	847	988	1130
60 "	"	"	94	118	176	235	353	471	588	706	824	942
70 "	"	"	81	101	151	202	303	403	505	606	706	807
80 "	"	"	71	88	132	176	265	353	441	530	618	706
100 "	"	"	56	71	106	141	212	282	353	424	494	565

The efficiency is calculated at 75 per cent.

Fig. 143 illustrates a form of steam-engine made by Hayward Tyler & Co., of London, much approved for situations where the machinery has to be conveyed a long distance overland to the mine.

FIG. 143.



STEAM-ENGINE ON WROUGHT-IRON BED PLATES.

The bed-plates are made entirely of wrought-iron, and can be taken to pieces for transport. The illustration is taken from an engine built for South African mining work. These engines are also made in pairs, either high pressure, condensing, or compound.

Raising and Transporting Ore.—The great improvements introduced in recent years in the manufacture of wire ropes have much extended their use, and conferred substantial benefit on mining industries, which in many cases have thus had requirements fulfilled which could not otherwise have been advantageously carried out.

Not only have wire ropes, as applied to haulage, added to the economy and efficiency of working, but in certain cases (markedly where used for aerial tramways), they have enabled mining to be carried on profitably in unfavourable positions, where otherwise the operation must necessarily have resulted in loss.

The saving effected by their employment instead of animal power is very great, as may be judged from the following figures for haulage, which are the results of 12 months' working at a leading colliery in England. 840 tons of coal are hauled per diem, the cost per ton per mile for the different items being as follows: Ropes, $\cdot 16d.$; coal, $\cdot 13d.$; labour, $1\cdot 2d.$; or an actual working cost of $1\cdot 78d.$ per ton per mile.

As a means, too, of transmission of power, wire ropes are invaluable.

In their selection, the point to be kept in mind is their special adaptability for the work they have to perform, for the systems of manufacture and qualities of metal used are various, one quality or

make suiting one condition of work, and another quality or make being adapted for another condition.

The usual qualities of steel used in the manufacture of wire of which ropes are made are technically known in the trade as—

Improved plough steel, having a breaking strain of 115 to 125 tons per sq. in.				
Plough steel	„	„	105 to 115	„
Patent crucible steel	„	„	80 to 90	„
Crucible steel	„	„	60 to 70	„
Bessemer steel	„	„	45 to 50	„

These names are sometimes fancifully altered, and we hear of “Best Improved Special Patent Crucible,” and so on, but these nomenclatural embellishments are simply for the purpose of making the wares more tempting.

Improved Plough Steel is found advantageous where the lightness of the rope is a *sine qua non*, as in the case of a very deep shaft and shortness of engine power; but Patent Crucible Steel is the quality most in vogue, as being more uniform in character. Charcoal iron is also employed, and where the rope has to run through bad water or corrosive substances, is used with great benefit.

It is well to bear in mind the fact that wire hardens in work, and that it is therefore an advantage not to use a wire too highly tempered, for although it will give a high breaking strain it is apt to get brittle.

It is apparent that where ropes have to bend to the curves of wheels or drums over which they have to work, the outer fibres of each wire, as they accommodate themselves to such curves, are in tension, and the inner fibres in compression, whilst the centre or neutral axis remains unchanged. Consequently it may be taken that the more flexible a rope (i. e. the less resistance it offers in the way of compression and tension in each wire) where it is subjected to much bending in work, the better will be the results, provided always that such flexibility be not obtained by the use of wire of such fineness that the wearing capacity of the rope is prejudicially affected thereby.

The difficulty of meeting these conditions will be at once seen, for in cases where only small drums and pulleys are admissible, and yet excessive friction has to be encountered, it is obvious that small round wires such as would be employed in an ordinary rope for such a purpose, would soon get worn through, whilst heavy wires would be speedily fractured, owing to the intermittent compression and elongation of their fibres, and which is necessarily greater in large than small wires on a given curve. Obviously too, a rope which would do admirably well as a fixed rope for aerial tramways, or as cage guides in shafts, where a certain amount of rigidity is advantageous, would be unsuitable for winding, hauling, or transmission of power where flexibility is essential.

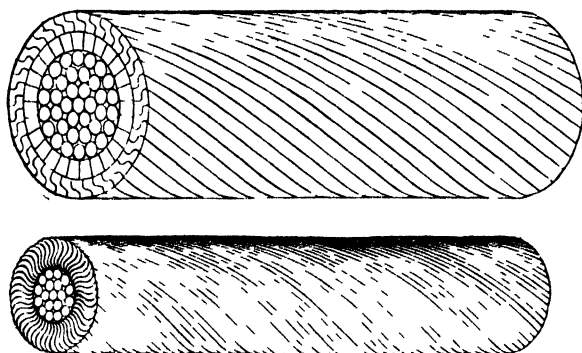
After these remarks, it is almost superfluous to say that it is always advisable to use the largest practicable drums, sheaves, or pulleys.

Great care should be taken to keep wire ropes well greased. A good preservative is a compound of vegetable tar, P.Y.C. tallow, and fresh slaked lime. It is of the first importance that any lubricant used be entirely free from acid.

There are many makes of rope, of which the most ordinary is that which consists of a wire or hemp centre around which are laid six strands composed of varying numbers of wires.

This style of rope has many objectionable features, to meet which Messrs. Latch and Batchelor invented in 1885 what is known in the trade as locked wire rope, and of which Figs. 144 and 145 are examples.

FIGS. 144 AND 145.



LOCKED WIRE ROPES.

Ropes constructed according to this system are coming greatly into use, and are giving exceedingly good results, in some cases, where great friction and bending take place, having lasted five times as long as the average of ropes supplied by various makers, and worked under the same conditions, and are still running apparently as good as when new.

Greater safety is claimed for the locked wire ropes, by reason of the inside coils, which constitute half the strength, not being exposed to wear. In ordinary ropes the inside wires appear in regular sequence on the outside of the rope, and once they are worn through or fractured the strength of the rope is gone.

Great flexibility can be imparted to the locked wire ropes without in the least impairing their ability to withstand excessive friction, as they have a wearing surface like a round bar.

They are also absolutely free from all tendency to spin, which to those who have experienced the sensation of being lowered in a bucket is a quality to be fully appreciated.

These ropes, too, are lighter, strength for strength, than ropes of any other make, and occupy less room and save a vast amount of wear upon pulleys or sheaves.

The interlocking of the wires on the outer cover prevents any wire, in case of fracture, rising from its place, and also prevents the intrusion of water or other corroding influences.

The following comparative table will be found useful.

Locked wire rope made of patent crucible steel.		Ordinary round wire rope made of patent crucible steel.			Tarred hemp rope.		Working load for inclines $\frac{1}{4}$ th of breaking strain		Working load for shafts $\frac{1}{4}$ th of breaking strain.		Breaking strain.
Circumference in inches.	Weight per fathom.	Circumference in inches.	Weight per fathom rope made entirely of wire.	Weight per fathom rope made with hemp centre core.	Circumference in inches.	Weight per fathom.					
	lb.		lb.	lb.		lb.	tons	cwts.	tons	cwts.	tons.
5	34	6 $\frac{1}{2}$	44	38	22	121	19	13	14	15	118
4 $\frac{1}{2}$	31	6 $\frac{1}{4}$	40	35	21	110	18	6	13	15	110
4 $\frac{3}{4}$	28	6	37	32	20	100	16	13	12	10	100
4 $\frac{1}{2}$	25	5 $\frac{1}{2}$	30	26	19	91	14	3	10	12	85
4	22	5	25 $\frac{1}{2}$	23	18	81	12	10	9	7	75
3 $\frac{3}{4}$	20	4 $\frac{3}{4}$	24	22	17	72	10	16	8	2	65
3 $\frac{1}{2}$	17 $\frac{1}{2}$	4 $\frac{1}{2}$	22	20	15	56	10	0	7	10	60
3 $\frac{1}{4}$	15 $\frac{1}{2}$	4	18 $\frac{1}{2}$	16 $\frac{1}{2}$	14	48	8	6	6	5	50
3	13 $\frac{1}{2}$	3 $\frac{1}{2}$	18 $\frac{1}{2}$	15 $\frac{1}{2}$	13	42	6	16	5	2	41
2 $\frac{3}{4}$	11	3 $\frac{1}{4}$	17	14 $\frac{1}{2}$	12	36	6	6	4	15	38
2 $\frac{1}{2}$	8 $\frac{1}{2}$	3 $\frac{1}{4}$	10 $\frac{3}{4}$	9 $\frac{1}{2}$	11	31	5	0	3	15	30
2 $\frac{1}{4}$	7	3	9 $\frac{1}{2}$	8	10	25	4	3	3	2	25
2	5 $\frac{1}{2}$	2 $\frac{3}{4}$	7	6 $\frac{1}{2}$	9	20	3	3	2	7	19

Some authorities deem that $\frac{1}{4}$ th of the breaking strain is sufficient for locked wire ropes in shafts, and $\frac{1}{5}$ th for inclines, the reason given being that there is not that rapid diminution of strength which takes place in ordinary ropes.

The use of wire tramways for the carriage of gold and silver ores has greatly extended, and the reason of this no doubt is, that the mines are, as a rule, placed in more or less inaccessible spots. The quantity to be carried from such mines is usually small, owing to the value of the mineral dealt with, and wire tramways of a light construction have therefore proved to be a most efficient means of transport. The annexed engravings show some examples of wire tramways thus applied.

In Fig. 146 the wire tramway system is shown applied to a mine worked in a mountain side at some thousand feet above the site of the mill; this distance is traversed by means of two self-acting inclines, one having a span of 1000 yd. without support, and the other a similar span

FIG. 146.



SELF-ACTING INCLINE WIRE TRAMWAY.

of 600 yd. without support. The loaded carrier, descending at a considerable velocity on a fixed rope, draws up the empty carrier on another fixed rope at a similar velocity, and by this means quantities from 50 to 200 tons per diem can be brought down at very little expense, the average cost of carriage with this system being for a length, say of a mile, about 2*d.* per ton. Ravines and valleys can be spanned on this system up to a width of 1200 yd., without intermediate support, and where the receiving end is sufficiently lower than the loading end, a self-acting system can be used. In Fig. 146 it will be seen that a distance of about one mile, with a fall of about 1000 ft., can be surmounted by two leaps, and the applicability of such a system in mountainous country can easily be understood.

For a different section of country, where the contour of the ground does not permit of long spans, the continuous running rope carried on posts as shown in Fig. 147 will be found most efficient. Lines of this system can be worked at from 1 to 3 miles in a section, and quantities conveyed from 50 to 500 tons a day, the motive power required for such lines being very small. The loads carried are proportionate to the quantity required to be transported per diem, and may be taken to vary from 100 lb. to 500 lb. This system is used for various purposes, and more than 200 miles of it are now in operation, a large portion of this being for mines. Water power is especially suitable for driving such tramways, being very steady and conveniently applied, owing to the slow speed at which the rope runs. In addition to the systems referred to, there are other special applications of wire ropes suitable for special situations, but these are too numerous to be referred to in detail.

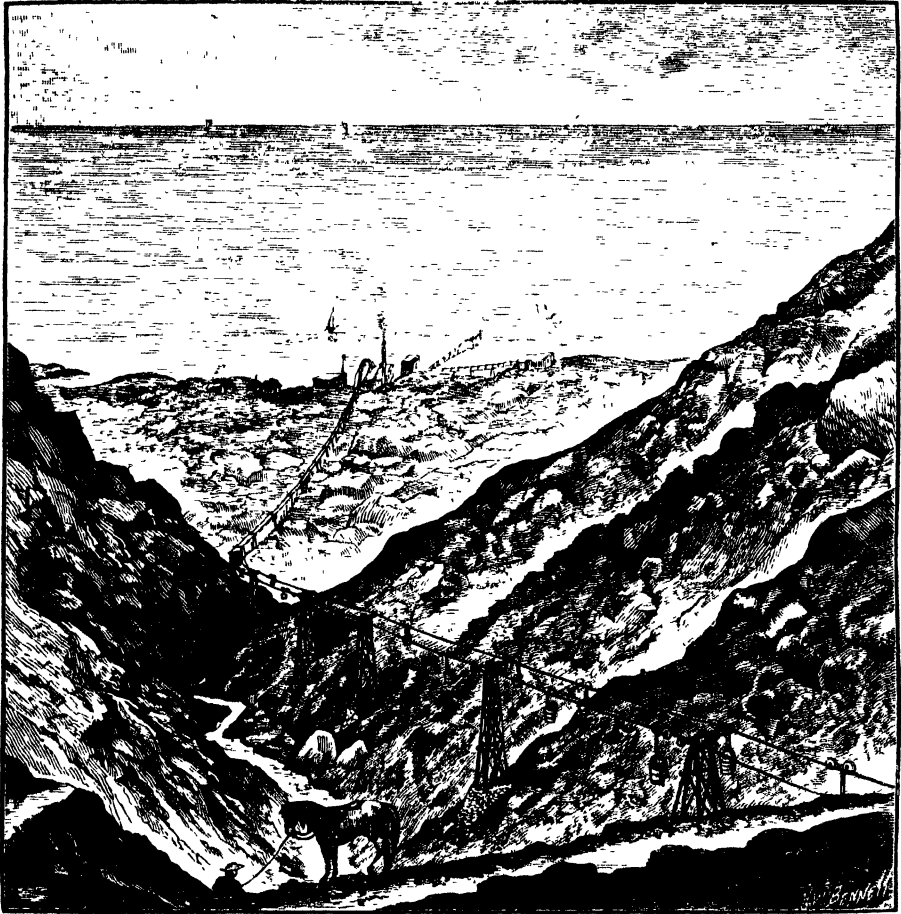
The cost of an ordinary wire tramway, such as shown in Fig. 147, may be taken at from 400*l.* per mile upwards, while that of the self-acting inclines depends a great deal on the spans required. As an illustration, a span of 1000 ft., capable of transporting 100 tons a day, in loads of 8 cwt., complete, ready for fixing, may be taken as costing about 500*l.* The adoption of this system of transport for mines is increasing daily, and on any but level ground almost certainly proves to be the most economical, both in first cost and in working. The illustrations above given refer to sections of wire tramways actually in work, which were constructed from the designs, and under the superintendence of, Mr. W. Carrington, M.Inst.C.E., 72, Mark Lane, London, by Messrs. Bullivant and Co., of the same address.

Wire tramways on the Otto system are being built in the Transvaal by Commans & Co., 52, Gracechurch Street, London.

In metalliferous mines the raising of the minerals is generally performed in a shaft or shafts specially devoted to that purpose. In collieries and other mines one shaft is often made to serve the

combined purpose of winding and pumping and ventilating, by dividing it into sections with what is called "brattice," or in other words, wooden partitions; but the general modern practice is to have two shafts, one forming the up-cast for the air, and the other the down-cast. Winding is frequently performed in both shafts.

FIG. 147.



CONTINUOUS RUNNING ROPE TRAMWAY.

The general principle of winding is now what it was in the year 1703, except that horse-power was then used and now steam.

A winding engine essentially consists of a drum, on which is coiled a rope, the two ends of the rope passing over pulleys supported on a head-gear over the shaft. To the end of the rope are attached cages, in

such a manner that when the drum is turned around, one cage ascends while the other descends.

When steam power was first introduced for winding, the engine ran at higher speed than the drum, and was connected to it by gearing. The speed of winding was slow, and where the shafts were not very deep the problem presented no special difficulties. The competition which has sprung up in mining, as in all other things, forced upon mine owners the necessity of economy. In establishments such as mines, the general charges are necessarily so great, that economy is best effected by increasing the output. The old geared engine was dispensed with (except in metalliferous mines, where the speed of winding is a matter of small consequence), and the direct-acting winding engine was substituted. The speed of winding was thereby immensely increased, not without certain mechanical difficulties. The weight of coal raised at a time was increased, necessitating the use of heavy wire ropes. When the cage was at the bottom of the shaft, the length of the rope in the shaft added greatly to the weight on the engine, so that the engine in starting had not only to overcome the weight of the coal, but also the weight of the rope. As the cage ascended, the weight of the rope decreased, while the weight of the rope attached to the descending cage increased until the cages met in the middle of the shaft, when the two ropes balanced each other. One of the earliest contrivances to overcome this difficulty consisted of a counter-balance by means of heavy chains attached to the opposite side of the drum. When the cage was at the bottom of the pit the chains were in their highest position, but as the cage ascended the chains descended into a shallow pit, resting themselves at the bottom. As the cage approached the top of the pit the chains were lifted again by coiling in the opposite direction on the drum. A counter-balance running on wheels on an inclined plane was sometimes used to produce the same result. A simple system of counter-balancing consists of a tail rope hanging from the bottom of the cages in a loop, extending to the bottom of the shaft, and there passing over a pulley. This arrangement virtually forms the winding rope into an endless band, the loop of which extends over a pulley at the bottom of the shaft. There is always the same weight of rope in the shaft, and the counter-balance is perfect. With this system, it is not necessary to coil the rope several times around the drum as is generally done, all that is required is to pass a loop of the rope over a sheave or pulley on the engine shaft, just like an ordinary driving belt. The friction between the rope and the pulley is sufficient to prevent slip, and to preserve the relative position of drum and cage. Winding is performed at very great speeds; it is therefore necessary that the engine-driver should have the reversing handle of the engine in his hand, with complete control over the engine movements. This condition

of things renders expansive working not an easy matter, and it is only within the last few years that it has been successfully introduced. The early expansion gears were too complicated. A simple gear essentially consists of a sleeve, which is moved longitudinally on the gear shaft of the engine by means of the reversing lever. On the sleeve are shaped cam surfaces for giving motion to the valves. The expansion cam will be understood if one imagines the old stepped cam, formerly used in marine engines, with the steps so numerous that they form a continuous curve. The longitudinal motion of this sleeve is under the direct control of the engine-man, by means of the reversing lever.

In increasing the speed of winding, mine engineers have not neglected considerations of safety, and winding cages are now provided with safety hooks; so that, should the engine-man accidentally overwind, the cage remains suspended in the headgear, instead of falling, as it would do without some such appliance, to the bottom of the shaft. Cages are also provided with safety appliances for causing them to be suspended in the guides, in the event of a rope breaking; but these are not very generally used. There is no reason why winding-ropes should break if properly inspected, and every care is now taken in that respect. Powerful steam breaks are applied to winding engines, so that, in case of accident, the engine-man can suddenly bring his engine to rest. In Germany it is the practice to employ mechanism attached to the engine, which shall cause the steam-brake to be automatically applied should overwinding occur.

Some of the most modern and improved patterns of winding and hoisting gears are manufactured by Ransomes, Sims, and Jefferies, Limited, Ipswich, who have for many years past made mining machinery one of their specialties, several of the gears at work in various mines in Mexico and the Cape having been specially designed by them.

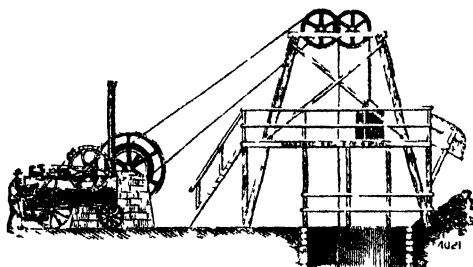
Fig. 148 represents a general arrangement of winding machinery, with staging and overhead pulleys, working in connection with a portable engine. This is a light kind of gear—specially adapted for prospecting purposes, or opening up new mines prior to putting down the main machinery.

The arrangement of winding gear represented in Fig. 149 is used generally with semi-portable or semi-fixed engines of the "underneath" type. It is made with single or double winding drum as desired, and frequently a crank is attached to the end of the drum shaft so as to adapt it to pumping purposes. The engines have steam jacketed cylinders, and the fire-boxes can be arranged for burning any description of fuel. The illustration represents the winding drums in connection with a semi-portable engine, but as before mentioned the same arrangement may be used with engines of the "underneath" type. The winding

drums in this and the other gears described are fitted with a powerful brake, and in some instances, as an extra precaution, an additional brake is fitted to the fly-wheel of the engine. The levers for the engine, stop-valve, reversing gear, and brake are placed close together, so as to be under the immediate control of the driver.

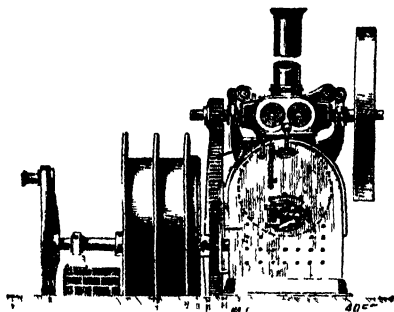
Fig. 150 illustrates a "steel-gearred" winding engine specially designed

FIG. 148



GENERAL ARRANGMENT OF WINDING MACHINERY.

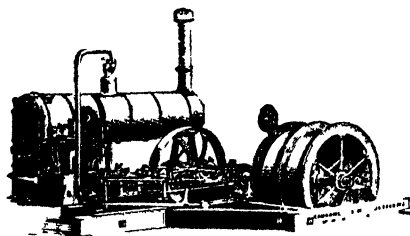
FIG. 149.



WINDING AND PUMPING MACHINERY

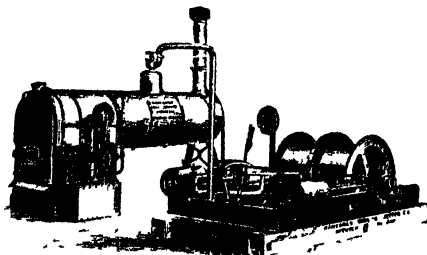
by Ransomes, Sims, & Jefferies. The boiler, it will be seen, is distinct from the engine, and this system presents the advantage that the weight of the packages for transport can be reduced to the greatest possible extent, making it specially suitable for districts where inland freight is expensive. The winding drums are driven by a pinion on the end of the engine crank-shaft, working into a spur-wheel attached to the side

FIG. 150



STEEL-GEARED WINDING ENGINE.

FIG. 151.



DIRECT-ACTING WINDING ENGINE.

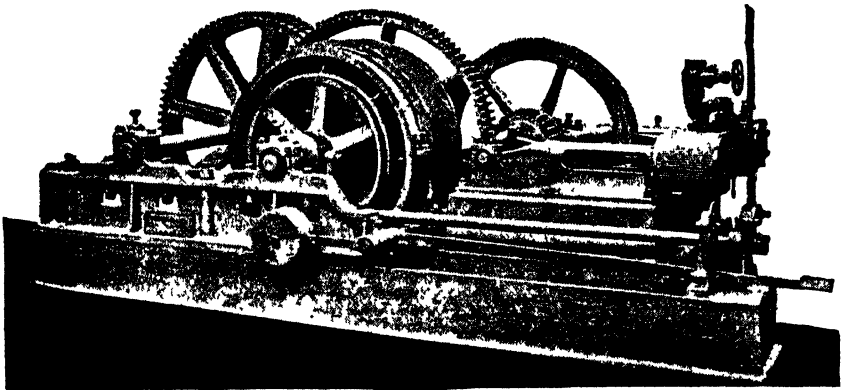
of the winding drum. Both engine and gear can be readily mounted on a stiff timber bed, thus dispensing with any other kind of foundation, a matter of great importance in outlying districts. This gear is adapted for raising heavy loads at, of course, a correspondingly slow speed.

Fig. 151 represents a "direct-acting" winding engine in which the intermediate gearing is dispensed with, the engine working direct on to

the drum shaft. This system, which possesses all the advantages of the gear described above, is intended for hauling light loads at a quick speed.

Fig. 152, photographed from a pair of engines built by Hayward Tyler & Co., for the South Indian Trevelyan Gold Mining Company, shows a very useful form of winding engine for trial shafts or for small workings. They are built with single or double winding drums. In addition to the winding engine there is a second shaft driven by gearing

FIG. 152.



COMBINED WINDING AND PUMPING ENGINE.

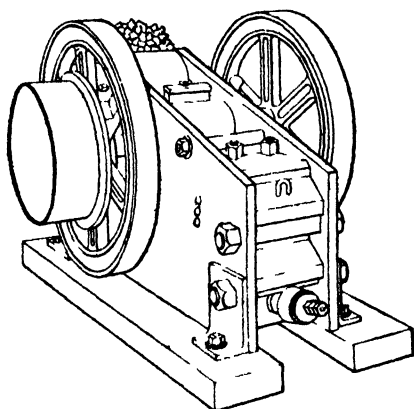
which may be disconnected at pleasure. This shaft is for driving a Cornish pump in the mine, by a T bob arrangement, somewhat similar to that shown in Fig. 133. The winding engine may of course be made without this addition.

TREATMENT OF THE VEINSTUFF.

Gold occurs in its matrix (the veinstuff) in a finely divided state and distributed in irregular proportions, both in a "free" condition (i.e. not associated with other metals or metallic salts), and in combination (of a most intimate mechanical nature, if not actually chemical) with other metals or their ores. The first step in effecting the extraction of the gold from the veinstuff is to bring about its disengagement from its associated mineral matters: firstly, by mechanical means, or disintegrating the mass to such a degree of fineness as will enable the gold and other metallic bodies to be liberated from the un-metalliferous ingredients; and secondly, by processes which will be described in a later chapter, separating the gold from any base metal or ore accompanying it. The former operation, here to be discussed, is termed "milling" in America, and is conducted in an establishment generally known as a reduction works.

Crushing.—The work accomplished by the “stamps” is very much facilitated and increased by subjecting the mineral as it comes from the mine (commonly called “stone,” “rock,” or “ore”) to the action of a crusher, by which it is reduced to a more convenient size for feeding into the “stamps.” Crushers are of many forms.

FIG. 153

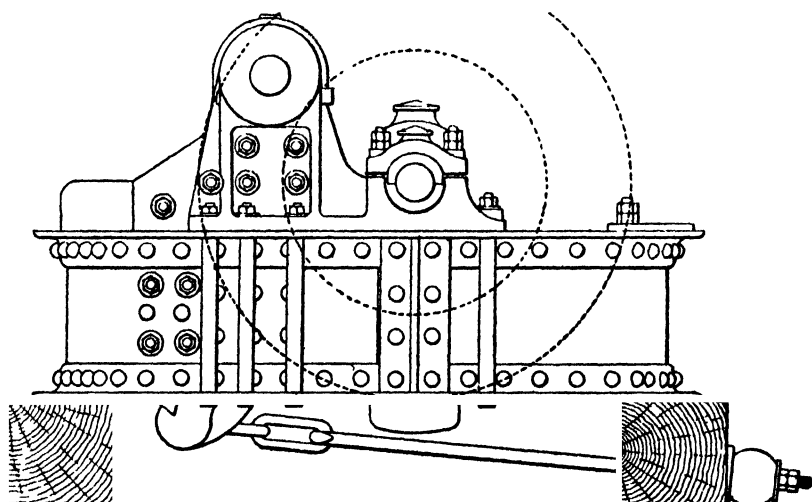


ROBEY'S CRUSHER.

Fig. 153 illustrates a very simple form of crusher, made by Robey & Co., Lincoln. To render its transport easy over bad or mountainous roads, steel is largely used in its construction, replacing the usual heavy cast-iron portions which add so much to the weight. The parts are few and simple, and afford access for oiling, cleaning, and adjusting, thus minimising the risks of breakage.

Fig. 154 shows an approved form

FIG. 154.



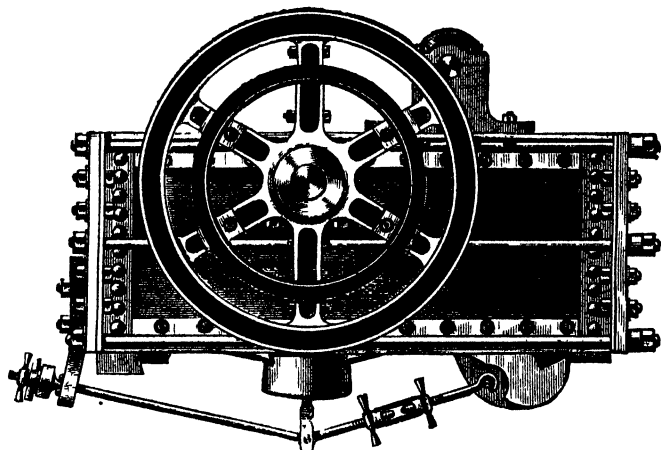
CALVERT AND CO.'S ORE CRUSHER.

of ore breaker, made by Calvert, Cornes, & Harris, 30, Walbrook, London.

Fig. 155 is an illustration of the type of ore crusher manufactured by

Robert Broadbent & Sons, Phoenix Iron Works, Stalybridge. These machines are specially designed for use in countries where there is exceptional difficulty or expense in transporting heavy weights. They are so constructed that they are readily put together by unskilled labour on arrival at their destination, and, being built up of steel plates and angle-iron, give the maximum of strength with the minimum of weight. They are so designed that there is no fear of the bolts working loose; and are fitted with Broadbent's patent drawback motion, by which a great saving in power is effected. In ordinary crushers, the movable jaw, when forced forward, has to compress a strong spiral spring embedded in rubber. This takes considerable power, and acts as a break to the machine. By Broadbent's positive drawback motion this is obviated; no unnecessary strain is put on the machine; the whole

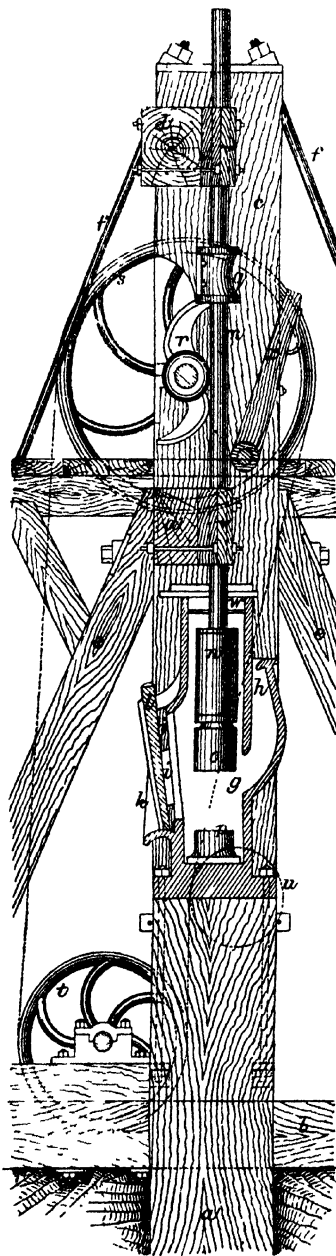
FIG. 155



BROADBENT'S ORE-CRUSHER.

power of the engine is available in breaking the material, as the jaw is pulled back by an arrangement of simple levers that cannot get out of order, which requires no power to work it, needs only adjusting for different lengths of toggles, and takes all strain off the eccentric shaft. The steel cushions or liners with grooves, in which the toggles are held, have hitherto been inserted from the side into dovetails, cast or formed in the parts to which they are applied. In this arrangement they are insufficiently secured against shifting laterally, and consequently they frequently work out at one end while the machine is in action, or at both ends if they break or split at the middle, as often happens in actual work; when this occurs, the cushions rub against the sides of the machine frame, absorbing a great amount of power, and forming ruts or grooves

FIG. 156.



CONSTRUCTION OF BATTERY.

in the frame sides, which thus become weakened, and breakages frequently result. By the use of Broadbent's safety steel toggle cushions this danger is entirely avoided. All the working parts of the machines, such as jaws, toggles, steel liners, &c., are renewable at small cost and without the aid of skilled labour. The different parts of Broadbent's machines are well balanced, the result being a crusher that invariably gives satisfaction. The machines can be fitted with feet or bolted on to barks of timber, as may be most convenient.

STAMPING.—The further comminution of the mineral is performed usually by "batteries" or "sets" (any number up to 6, but generally 5) of "stamps," which are heavy iron pestles, lifted to a height of some inches (say 7 to 15 in.), and allowed to fall by their own weight upon the mineral intended to be powdered. These stamps work in a "mortar" or "coffer," an iron trough, constantly and regularly supplied with ore and water, and from which the crushed material escapes, as soon as it is reduced to the desired degree of fineness, in the form of a liquid mud or "pulp," through the meshes of "screens" or "gratings" closely fitted in the sides of the "mortar." This latter is generally of rectangular form, resting on a solid foundation, and established in a substantial timber framework. The stamps are successively lifted at regular intervals by means of revolving "cams" or "wipers," arms of iron keyed to a cam-shaft, placed directly in front of the battery, and receiving its motion from the driving-power of the mill. The stamps move vertically between guides that form part of the battery-frame

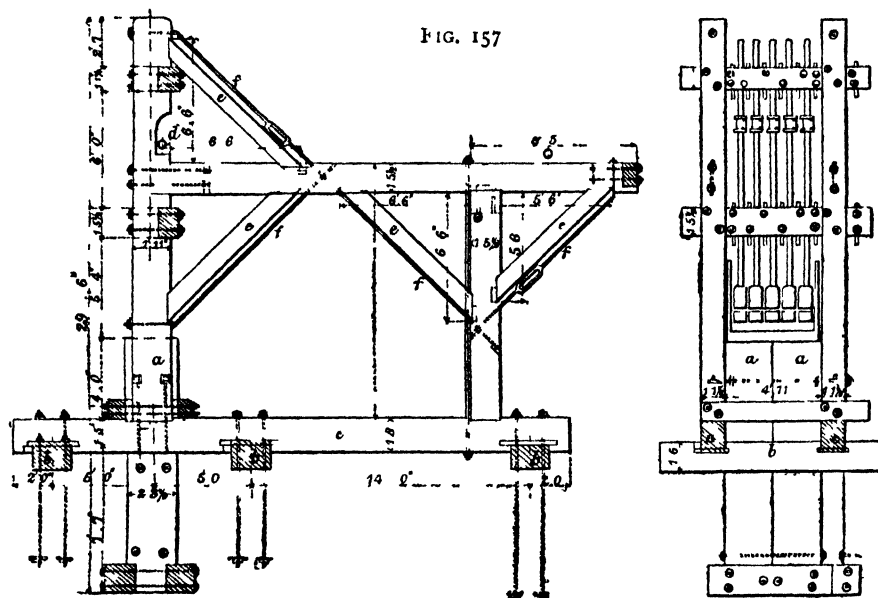
The general construction of the several parts of the battery is shown in Fig. 156 (scale $\frac{3}{8}$ in. = 1 ft.):—*a*, foundation timber or mortar-block; *b*, transverse sill; *c*, battery-posts; *d*, tie-timbers; *e*, braces; *f*, tie-rods; *g*, mortar; *h*, feed-aperture; *i*, screen or grating; *j*, screen-frame; *k*, lugs to secure frame; *l*, wedge or key; *m*, stamp-stem or lifter; *n*, stamp-head; *o*, shoe; *p*, die; *q*, tappet; *r*, cam; *s*, pulley on cam-shaft; *t*, driving-pulley; *u*, tightener; *v*, guides; *w*, battery-covers; *x*, prop for supporting stamp when not at work.

Foundations.—The foundation-timber or mortar-block for batteries of this character often consists of heavy vertical timbers, placed close together, and firmly connected by cross-timbers and iron bolts. The timbers may be 6 to 12 ft. long, according to the nature of the ground and the proposed height of discharge from the mortar. Sometimes the timbers are laid horizontally, so as to serve as the base of two or more batteries. When the foundation-timbers are in place, the space about them is packed and stamped as firmly as possible with clay or earth. When the ground on which the batteries are to be built is a hard compact gravel, or a firm clayey material, the surface is sometimes levelled off so as to admit of laying the transverse sill-timber *b* of the battery-frame, and a narrow pit is then excavated, some 6 to 14 ft. deep, and long and wide enough to receive the ends of the mortar-blocks; the posts or blocks are introduced into the pit in a vertical position, their bottom ends resting directly on the ground without any intervening horizontal timber. The remaining space in the pit is then compactly filled with clay, which is pounded or stamped firmly into place. The sill-timbers *b* and battery-posts *c* are securely bolted to the foundation-timbers. The posts *c* are braced by the timbers *e* and rods *f*, and are connected by the tie-timbers *d*, which also support guides *v*.

It must ever be borne in mind that the foundations are of prime consequence. When improperly constructed, the battery cannot be run at its full speed and capacity, without shaking itself to pieces, whereby great delay, expense, and actual loss of metal are sure to arise. Extra care in securing complete solidity for the battery in the first instance will be amply repaid, while nothing will compensate for a rickety structure. Often it is advisable to excavate the foundations down to the solid rock, where that is not more than 14 ft. below the surface. At some mills, the trench itself is cut in the solid bed-rock, leaving about 2 ft. all round for packing.

Fig. 157 shows the details of the foundations and framing more minutely. The mortar-blocks *a* are 30 in. square, and 12 to 14 ft. long. They are made quite true, and thoroughly coated with Stockholm tar, applied hot, then bolted together by $1\frac{1}{2}$ -in. pins and nuts. The transverse sills or foot-timbers are square, 6 ft. long, and are let 6 in.

into the mortar-block, freely tarred, and bolted together by six $1\frac{1}{8}$ -in. pins after being squared. At 5 ft. from the top, the mortar-blocks are cut to 59 in by 29 in. The prepared blocks are let down upon the floor, and levelled up by putting sand beneath. When in place, the height is



DETAILS OF FOUNDATIONS AND FRAMES

accurately determined, and a level is run across the whole set. The tops are planed smooth and dished about $\frac{1}{16}$ in., to prevent the surface becoming rounded; they are kept covered till the mortar is fixed on the top.

Frames.—Battery-frames in America are usually made of the best red spruce (*Abies rubra*) or sugar-pine (*Pinus Lambertiana*). First, three battery-sills *b* (Fig. 157), 18 in. by 24 in., and 28 ft. long, are placed parallel to the direction of the cam-shaft, one being 5 ft. from centre to centre behind the mortar-block, a second 5 ft. in front, and the third 14 ft from the second. They are secured by bolts 8 ft. long, keyed into the masonry or to the bed-rock. In the latter case, holes 3 ft. deep and $1\frac{1}{2}$ in. diameter are bored in the rock; the bolts are slotted at 6 in. from the lower end, and wrought-iron wedges, $\frac{5}{8}$ in. by 1 in., 5 in. long, with a head 1 in. square, are made to fit the slots; the bolts are inserted in the holes, and driven so that the wedges enter up to their heads, when the holes are filled up with molten brimstone. Cast-iron washers and nuts retain the bolts in the sills. Next, the outside-line timbers *c*, measuring 20 in. by 14 in., and 28 ft. long, are wedged into the battery-sills, and

secured by bolts. The top of the sill should be 4 ft. above the top of the mortar-block. The centre-line timbers measure 20 in. square and 28 ft. long. The intermediate-line timbers measure 20 in. by 14 in. by 28 ft., and are dressed on the upper side and reduced to $13\frac{1}{2}$ in. and $19\frac{1}{2}$ in. where they pass the battery-blocks; they are let 3 in. into the sills, and are secured by keys driven both ways and by two iron bolts 33 in. by $1\frac{1}{8}$ in. The outside battery-post measures 23 in. by $13\frac{1}{2}$ in., and is tarred and let into the sills. The posts for as many as four batteries may be raised simultaneously. The middle one is usually of somewhat larger scantling, say 23 in. by $19\frac{1}{2}$ in. The posts are secured to the line-timbers by two 1-in. joint-bolts, 44 in. long. In the upper part of the posts is cut the cam-shaft journal-seat *d*. The posts are held together by the tie-timbers *d* (Fig. 156), carrying the stamp-guides *v*. The bracing and tie-bars *e f* are all arranged to ensure the greatest possible steadiness during work. A travelling block and tackle suspended over the battery will be found very useful for inserting and removing the stamps.

Mortars.—Mortars are often fixed directly upon vertical mortar-blocks, without any horizontal sill intervening. When the frame is ready, the temporary covering is removed from the mortar-block, and the holes for the mortar-rods are bored from the template taken from the bottom of the mortar. All cracks in the block are filled up with molten brimstone, and its surface is again planed and tarred. In the Western States of America, it is a favourite custom, before fixing the mortar, to cover the top of the block with a triple thickness of common domestic blanket, thoroughly tarred on both sides. Upon this cushion, the mortar is bolted with $1\frac{1}{2}$ -in. pins *d* (Fig. 158). This plan reduces the "jar" to a minimum, and prevents the gradual loosening of the mortar from the block, and consequent introduction of material into the space, whereby the perfect level of the mortar is destroyed. The rate of discharge is partly governed by the width of the mortar, increasing as the mortar narrows; but if the ore is very hard, it will cause frequent renewals of screens should the mortar be very narrow.

Mortars may be constructed partly of wood and partly of iron—the sides and ends being of wood, and the bed-plate of solid iron—or they may be entirely of iron. The latter plan is now general, as with the compound method there is great trouble in keeping tight joints. The form commonly adopted is an iron box or trough, 4 or 5 ft. long and deep, and 12 in. wide inside, preferably cast in one piece, but sometimes made in sections bolting together, where transport is difficult. The bottom is always made very thick, as it has to bear the chief strain; but in positions remote from iron-foundries, it is an advantage to have the sides cast thin, and to attach a lining which can be renewed at will. This form of mortar is shown in Fig. 159. The feed-opening *a* is an aperture 3 or 4 in. wide,

and nearly as long as the mortar, by means of which the ore, suitably sized, is fed into the stamps. On the opposite side is the discharge, furnished with a screen *b*, by which the "pulped" material escapes; this opening is almost as long as the mortar, and 12 to 18 in. deep, the lower edge being 2 or 3 in. above the top of the die *c*. The bolts *d* hold the mortar on the block *e*.

The Californian high mortar varies in weight from 3000 to 6000 lb.; it is usually about 4 ft. 7 in. long, 4 ft. 2 in. to 4 ft. 4 in. high, 12 in. wide inside where the dies are set, and 3 to 6 in. thick in the bottom. Mortars which are made in sections are termed "section-mortars." The one illustrated in Fig. 159 is 4 ft. long, and will take five stamps. The upper

FIG. 158

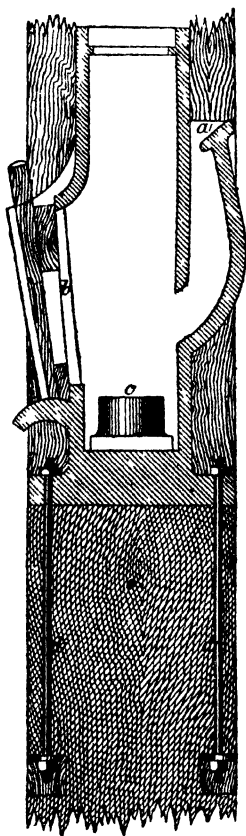


FIG. 159.

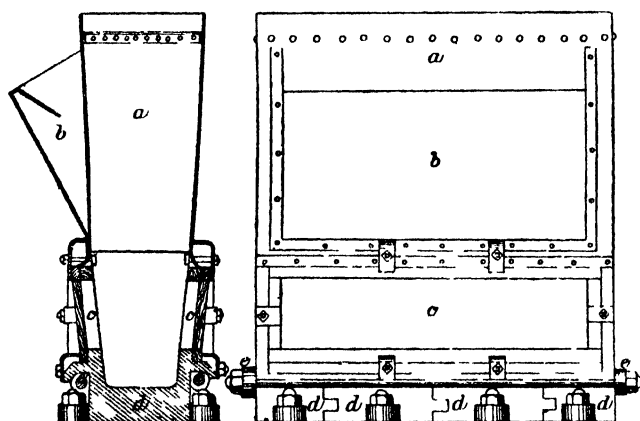
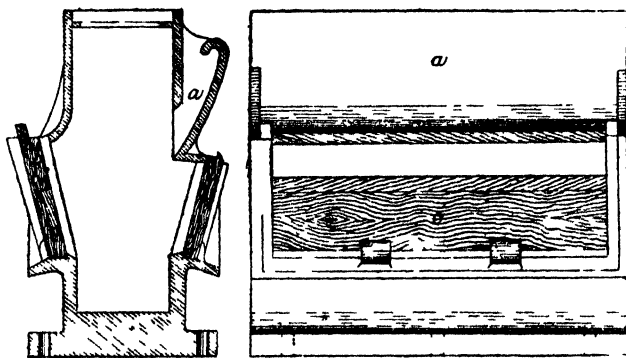


FIG. 160



MORTARS OR COFFERS OF VARIOUS FORMS.

portions *a* are of boiler-plate, strengthened with angle-iron; the feed-opening is at *b*; there is a double discharge with screens at *c*, the screens

being attached by movable lugs or clamps ; the bottom is cast in four sections *d*, which are accurately fitted together with tongued and grooved joints, planed, and held by heavy iron bolts *e* running through them from end to end, and secured by strong nuts on the outside.

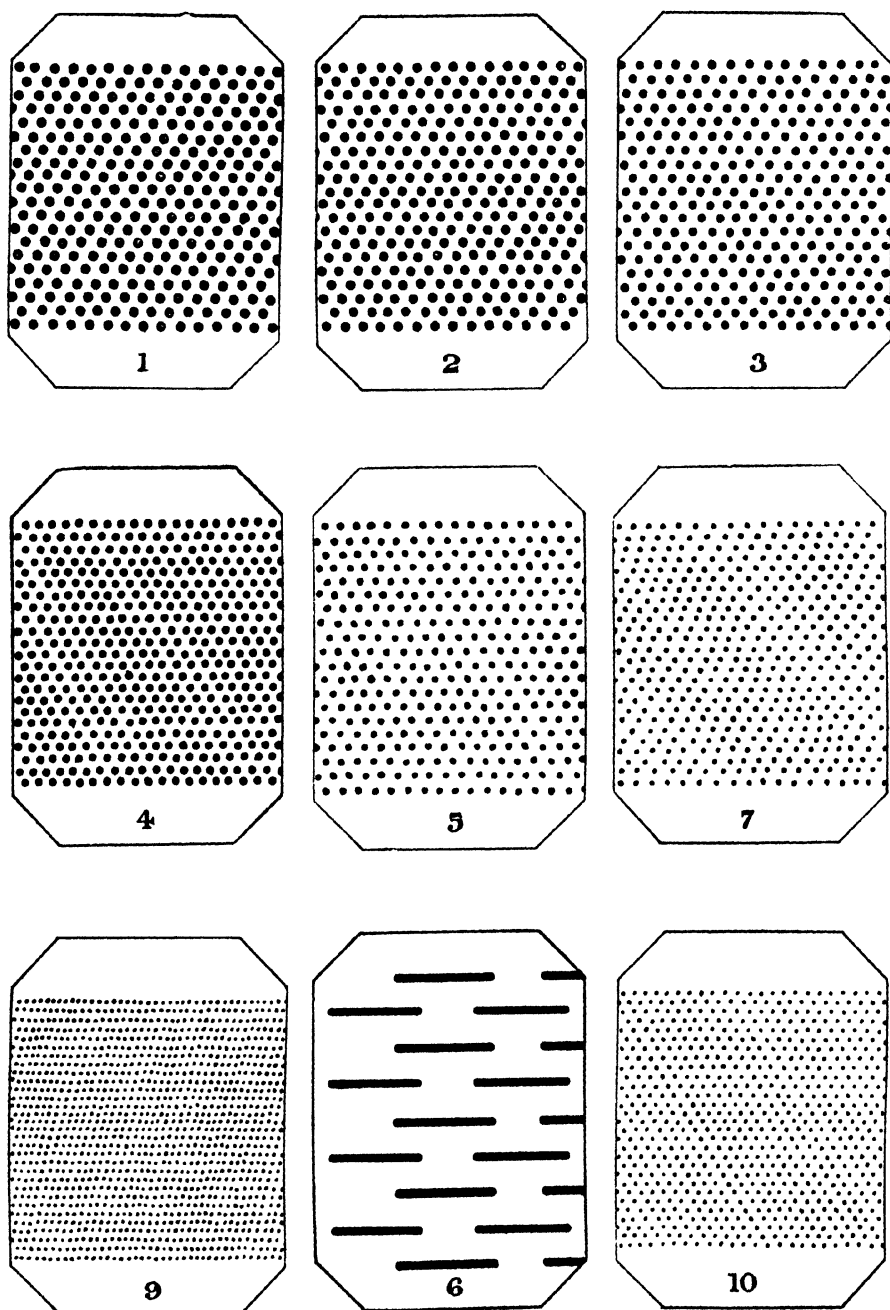
Donnell's mortar, much used in gold-ore crushing, is shown in Fig. 160. The ore is fed in at *a*, and the discharge is at back and front. The screen *b* is narrow, placed high above the dies, and occupies only a part of the opening in front. The lower portion of this opening, and the opening at the back, is closed by a wooden door *c*, covered on the inside by a sheet of amalgamated copper.

Screens.—The action of the stamps, when properly supplied with ore and water, results in the reduction of the solid matters to such a degree of fineness as will enable them to flow off with the water, which wells and splashes up, at each blow of the stamps, through the screens or gratings placed at the exit from the mortar. The position of the screens in the mortar has been already shown in Fig. 156. The screen should incline outwards at the top, to facilitate the passage of the pulp through its meshes. The length of the screens will vary with the length of the mortar, and the width is usually 10 to 15 in.

The shape, disposition, and size of the orifices in the screens are subject to the greatest possible variety. A few of the many patterns in use are shown in Figs. 161 and 162. The "gauge," or number of holes per sq. in. adopted in Victoria, ranges between 60 and 800 ; in America, it runs from 900 to 10,000. When the holes are round, their size is graduated by the numbers of sewing-machine needles, from 0 to 10 : thus No. 5 is about $\frac{1}{16}$ in. diameter, and No. 8 about $\frac{1}{8}$ in. When the holes are slots, they are usually $\frac{3}{8}$ in. long and of the same diameter as a No. 6 needle. As regards material used in the construction of screens, Americans are universally in favour of Russia sheet iron or sheet steel, $\frac{1}{32}$ in. thick, weighing about 1 lb. per sq. ft., very soft and tough, with a clean smooth surface, and perfect freedom from rust or flaws ; in Australia, sheet copper is often employed, that at the Port Phillip works being $\frac{1}{16}$ in. thick, with 84 holes per in. The holes should always be punched. This operation leaves one side in a rough state, like the outside of a nutmeg-grater. The rough side is turned inwards, towards the wear of the issuing pulp ; and, as the inner end of the orifice is smaller than the outer, there can be no fear of the meshes becoming clogged, as everything which enters from the inside can more readily escape on the outside. A 5-stamp battery usually requires 13 sets of screens per annum, a set consisting of 5 sheets of 1 to $1\frac{1}{2}$ sq. ft.

The "gauges," or number of holes per sq. in., in the samples of screens shown in Figs. 161 and 162, are approximately as follows :—

FIG 161.

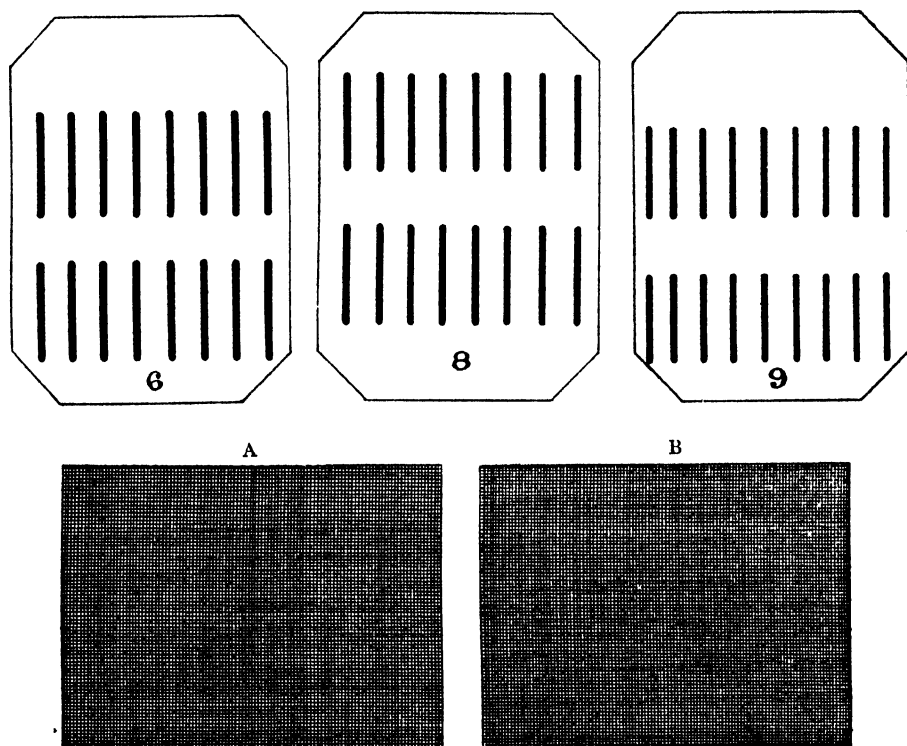


SCREENS OR GRATINGS (actual size)

No. 1, 150 holes per sq. in. ; Nos. 2 and 3, 140 ; No. 4, 200 ; No. 5, 160 ; No. 7, 220 ; No. 9, 575 ; No. 10, 320 ; A, 3600 ; B, 3250.

It is very essential that the material of which the screens are composed, whether stamped sheets or woven wire, should be of the best quality attainable, as the wear in any case is very rapid, and with inferior material the loss of much valuable mineral may result through the escape of particles not sufficiently reduced in size to liberate the gold. In this connection mention may be made of the well-known "agate steel,"

FIG. 162.



SCREENS OR GRATINGS (actual size).

stamp battery cloth, specially manufactured for gold mining by N. Greening and Sons, Limited, Warrington, England. Nothing can surpass their screens for regularity of mesh, and the heavy qualities are remarkably substantial and enduring.

Some mills adopt tinned iron, first burning off the tin ; being thinner than Russia iron they allow of more rapid discharge, but do not last so long. Brass wire screens endure 10-14 days ; Russia iron, 15-40 days. Common steel wire is liable to rust.

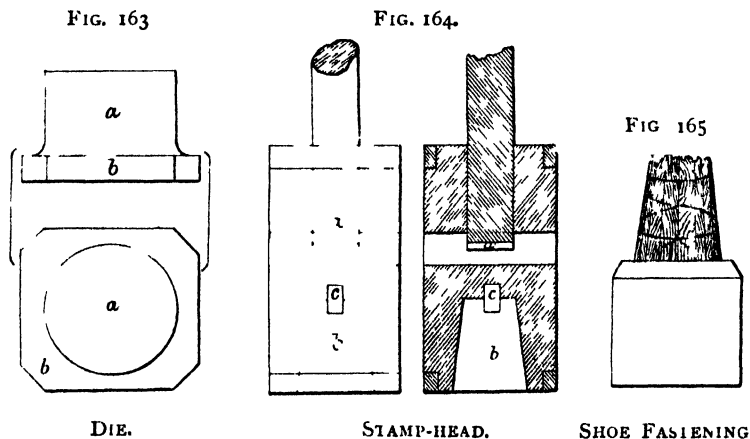
The "stamp duty" or effective capacity of a battery is governed entirely by the facility with which the pulp can escape. It is therefore obvious that proper adjustment and gauge of the screens are most important matters. The object sought is to reduce the mineral just sufficiently to enable the gold to free itself from the gangue, and to thus reduce the greatest possible quantity in a given time. Microscopical examination of the ore and of the pulp will afford some guide in this respect, as will also the same test applied to the tailings escaping from the mill. It is hardly necessary to point out that the rate of discharge will also be proportionate to the area of screen presented to the pulp, whence it follows that the screens should be as deep and wide as possible. The most rapid delivery would be attained by having a single stamp surrounded by screen on all sides; and every battery should at any rate have a back and front delivery. Where the feed is on one side and the discharge on the other, the working power of the battery is lessened by one-half. The importance of this is apparently not yet recognised by some engineers. Experiments have shown that pulp fed back into the mortar takes nearly as long to escape through the screens as rock which has first to be reduced to pulp. Fine stamping does not necessitate the use of fine-gauged screens, as the same result may be gained by elevating the screen, though with a certain loss of effective capacity.

Dies.—In order to save the mortars from wear and tear, "dies" or "false bottoms" are placed in them, to receive the blows from the stamps. In America, the die is a cylindrical piece of cast iron, corresponding in form to the shoe of the stamp that falls upon it, and 4 to 6 in. high. Some mortars are made with circular recesses in the bottom, for the dies to fit into. In others, to prevent the material from working in under the die and displacing it, the circular recess in the bed-plate is cast with a flange, and the die with a small projection or lug. A groove is also made in the bottom of the mortar, so that the die may be introduced, with its lugs dropping into the groove; the die being then turned about 90° , the lugs come under the flanges of the recess, and the die is thus held in place. A simpler and more general plan is to cast the die with an upper circular "boss" or die proper *a*, on a square flat "foot-plate" *b* (Fig. 163). The bottom of the mortar is then also made flat, and the dies are dropped in, resting on their foot-plates, which just fill up the space in the floor of the mortar. The corners of the foot-plates of the dies are bevelled off, so as to allow of the insertion of a pick for effecting their removal when necessary. The foot-plate of American dies is usually $1\frac{1}{2}$ to 2 in. thick and 10 to 12 in. square; the boss is 3 to $5\frac{1}{2}$ in. high and 8 to 10 in. in diameter. It is of hard tough cast iron, and is chilled down to the foot-plate. In Victoria, it is found to be a good plan to allow the dies to rest immediately upon a layer of finely-broken quartz, at least 3 in. deep, by

which means an opportunity is provided for the liberated gold particles to get into the gravel, out of reach of the stamps, and whence they can readily be recovered. In some localities, the die consists of a simple slab of iron filling the whole mortar, and which is turned over when one side is much worn; but the wear is liable to be uneven, and the die often breaks before it is worn out.

Stamps.—The stamp consists of the stem or lifter; the head or socket attached to the lower end of the stem, and furnished with a shoe; a movable part which sustains the force of the blows and the wear of the operation; and the collar or tappet, by means of which the revolving cam lifts the stamp for its fall. The stem is a round bar of wrought iron, about 3 in. in diameter, usually turned in a lathe. Its length is 10 or 12 ft. Its lower end is slightly tapered, and corresponds in form to a socket or conical hole in the upper part of the stamp-head. The rest of the stem is usually made round throughout its entire length, the method, now in general use, of attaching the tappets to the stems not requiring any modification in the form of the latter, as was formerly the case.

The stamp-head, illustrated in Fig. 164, is a cylindrical piece of tough cast iron, about 8 in. in diameter and 15 in. high. In its upper end is a socket, shown by dotted lines, corresponding with the axis of the cylinder, and conical in form, designed to receive the slightly tapering end of the stem, to the dimensions of which it must be adapted. This conical hole or socket is about 7 in. deep. At its bottom is a hole or key-way *a*,



passing through the head at right angles to the cylindrical axis, by which passage a key may be driven in to force the head from the stem when necessary.

To attach the stamp-head to the stem, the latter is placed in its position between its guides, the head standing immediately under it

The stem, being dropped, enters the socket, and a few blows of the hammer drive it in with sufficient force to cause the head to be raised when the stem is lifted. The stem and the head, being suffered to drop together a few times, become firmly connected. In the lower end of the head is a similar hole or socket *b*, but larger than the upper one, likewise tapering or conical in form, made to receive the stem or shank of the shoe, which is thus connected with the head in a similar manner; a rectangular hole or passage *c* through the head at the end of this lower socket, permits the removal of the shoe in the same way as the stamp-stem is forced out from the upper socket. A stout wrought-iron hoop encircles each end of the stamp-head, being fitted and driven on when hot, and allowed to shrink in place.

The shoe in common use is a cylindrical piece of cast iron about 8 in. in diameter and 6 in. high, above which is a shank or stem, the base of which is 4 or 5 in. in diameter, tapering in form, and about 5 in. high. It is made of the hardest white iron. It is attached to the head in manner somewhat similar to that just described for connecting the head and the stem, but is wedged on by means of strips of pine-wood. These strips, which are cut about as long as the stem of the shoe, $\frac{1}{4}$ in. thick and $\frac{1}{2}$ in. wide, are placed around the stem of the shoe, and tied with twine, as shown in Fig. 165. They must be thick enough to wedge the stem of the shoe firmly in its socket, without allowing the head to come into contact with the body of the shoe. When the shoe is ready to be fixed to the head, it is placed in proper position, with the stem of the shoe directly under the socket of the head, and the stamp and head are then allowed to drop upon it. If necessary, a few blows of a hammer are struck upon the top of the stamp-stem. The whole may then be raised, the shoe keeping its place, and suffered to fall repeatedly, until the shoe is firmly established in its socket. During this operation, a piece of plank is interposed between the die and the shoe, for the latter to strike upon. When a shoe is worn out, it can be removed from the socket by driving the key into the key-way *c*, and forcing it off. Care must be taken that the shoe does not become so thin as to permit the head to sustain undue wear, and so become weakened. Shoes should be renewed when worn down to a thickness of 1 in.

It was at one time objected that the effective capacity of round stamps was less than that of square ones; but it has been proved, by careful experiments under uniform conditions, that they are equal in this respect. At the same time, the circular stamp possesses a great advantage in that it can be caused to revolve on its own vertical axis while at work, making a partial revolution at each blow, the rotary motion being continued during the free fall of the stamp, which produces a grinding effect upon the material between the shoe and the die,

increasing the effective duty of the stamp, and equalising and reducing the wear of the shoe. Bland declares himself satisfied that the square heads used at the Port Phillip works are more efficient and economical, the cam-barrel being simpler and more easily kept in order. Round revolving stamps are almost a necessity when the feeding is done by hand from the back only ; but with self-feeding all round the stamps, he considers the square head the better.

Weight of stamps.—This is subject to great variation. In Victoria, the figures usually range between 224 lb. and 1232 lb. per head ; in America, 700 to 950 lb. Generally a medium weight of 560 to 672 lb. best suits the character of the ore, but some ores are met with requiring the higher figures.

Height of drop.—The height of the drop or fall of the stamps varies from 2 to 11 in. in Victoria, and from 7 to 11 in. in America. It should not be less than 7 in., and may be increased with advantage if the stamps are light.

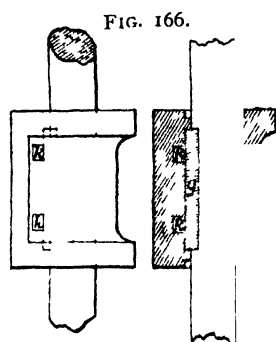
Speed.—The number of drops made by each stamp per minute is the "speed" of the battery. In Victoria, it varies from 45 to 85 blows, 75 to 80 being generally considered most effective ; in America, 70 to 100. The greater speed adopted in America is suited to the system of amalgamating in the battery.

Order of drop.—The order in which the stamps drop varies in different mills, but the desired conditions are (1) that the work of raising the stamps shall be uniformly distributed on the cam-shaft, so that the weight lifted shall be, as nearly as possible, the same at any period of the revolution ; and (2) that each stamp shall fall effectively upon the material to be crushed, and maintain its proper distribution in the mortar. If all the stamps fell at the same time, the structure would soon be knocked to pieces ; and if they fell in regular succession, from one end of the battery to the other, the material would accumulate at one end, and the effective duty of all the stamps would be greatly diminished. In a 5-stamp battery, the common sequence is 3, 5, 2, 4, 1, or, in other words : (1) the middle stamp, (2) the end one on the right, (3) the second on the left, (4) the second on the right, (5) the end one on the left. Another sequence, which makes a backward and forward wave, and thus keeps the mortar very evenly filled, is 3, 4, 5, 2, 1. Others which find favour are—3, 4, 2, 1, 5 ; 2, 4, 5, 3, 1 ; 3, 5, 1, 4, 2 ; 1, 5, 2, 4, 3 ; 1, 5, 4, 2, 3. It is thought that the middle stamp dropping first secures the greatest discharge, and that the end stamps dropping first effects the maximum of work done.

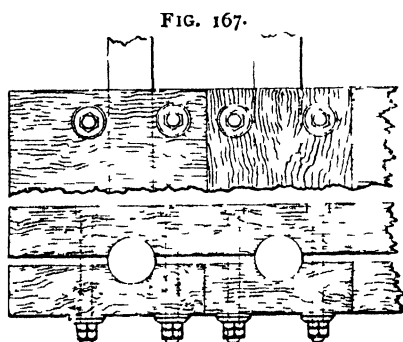
Character of blow.—The character of the blow delivered by the stamp upon the ore demands attention. The hardness of the mineral containing gold is almost always so much greater than that of the gold itself, or

even of auriferous pyrites, that the same amount of stamping on the three substances will render the two latter much finer than the former. But it is of the utmost importance to prevent the gold being smashed too fine, or beaten flat, for in those conditions it is very difficult to save effectively. The tendency of slow heavy blows is to flatten the gold-particles, while that of smart light blows is to effect disintegration without materially altering the shape of the particles.

Tappets.—The collar or tappet is a projecting piece firmly secured to the upper part of the stem of the stamp, by means of which the revolving cam may lift the stamp and let it fall upon the substance to be crushed. Tappets vary in form and method of attachment to the stem, but that which seems to combine the greatest number of advantages, and to have been most generally adopted on the Pacific coast, is that which is known as Wheeler's "gib-tappet." Fig. 166 shows an elevation and vertical section of this contrivance. It is a piece of cast iron, cylindrical in form,



TAPPET OR COLLAR.



STAMP-GUIDES.

about 8 in. in height and diameter, and hollow at the centre, so as to receive the stamp-stem. To secure the tappet to the stem, there is a gib *g*, about 2 in. wide, and nearly as long as the tappet, having its inside face curved so as to correspond in form to the circular hole through which the stem passes. The gib being fixed in its place in the tappet, and the latter being upon the stem, it is pressed against the stem by means of two keys *k*, driven into the key-ways with force sufficient to hold the tappet and stem firmly together, and prevent slipping between them. This is found to be a very effective method of securing the tappet, while permitting it to be fixed at any desired point on the stem, according to the wear of the shoe. The stem is uniform in size, and the work of cutting facings, screw-threads, and key-scots on the stem, required by other methods, is thus avoided. The revolving cam, meeting the tappet, and raising the stamp, causes it, while being lifted, to make a partial revolution about its vertical axis.

Guides.—The stamp is held vertically in its movement by guides, between which the stem passes. These were formerly made of iron, but such have been almost entirely replaced by wooden ones in Nevada and California. One set of guides is placed below the tappet, about 1 ft. above the top of the mortar; the other set is placed near the top of the stem, so that 6 in. or 1 ft. of the latter may project above the guides. They are supported by the cross-timbers or ties *d* (Fig. 167), which form a part of the battery-frame, connecting the two uprights or posts. They are usually made of pine, though hard wood is preferred, and are 10 to 16 in. wide. One part of the guide is made in a single piece for the whole battery, and bolted to the cross-timber; the other may be in one piece, like the first, or cut into as many pieces as there are stamps in the battery, as in Fig. 167, which are then secured to the corresponding part by bolts. In each part are cut semicircular recesses, which form, when the two parts are put together so that the recesses correspond, the holes or stem-ways for the reception of the stamp-stems. When the guides are so much worn by friction as to permit too much motion of the stems, they may be dressed down on their adjacent faces, by which means the recesses are reduced to nearly the proper dimensions.

Cams.—The cam is a curved arm fixed to a shaft, which is so placed in front of the battery that, by the revolution of the shaft, the cam is brought into contact with the tappet of the stamp-stem, causing the tappet to rise to a height determined by the length of the cam, and to fall at the moment of its release from such contact.

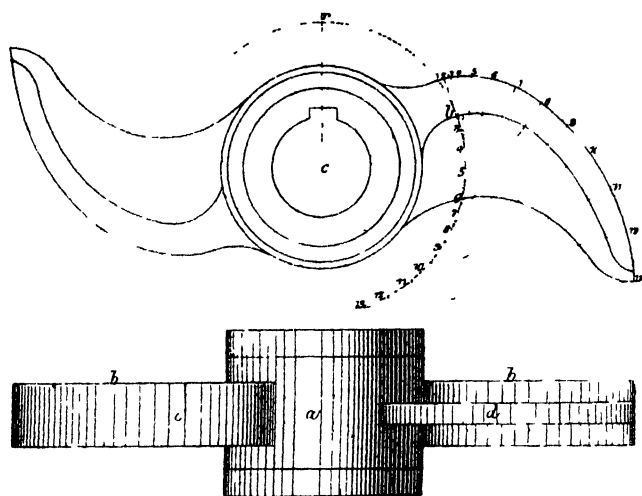
In Nevada, the cams are made of tough cast iron, and are usually "double-armed," that is, have two arms attached to one central hub. Fig. 168 shows the form of cam generally in use: *a* is the hub; *b*, the arms; *c*, the face; *d*, a strengthening rib. Where the speed of the stamps is great and the fall slight, single-armed cams are necessary, or the constantly recurring impact of the cam would virtually keep the stamp suspended, or at least so lessen its fall that it would not reach the material in the battery.

The proper curve of the face of the cam, in order that it may perform the required duty with the least friction, is the involute of a circle the radius of which is equal to the distance between the centre of the cam-shaft and the centre of the stamp-stem. This produces a line for the face of the cam which meets, better than any other, the various requirements. The bottom of the tappet is constantly perpendicular to the radius of the curve of the cam; the tappet, and with it the stamp, is lifted vertically and uniformly, so that the lift of the stamp is always regularly proportioned to the revolution of the cam-shaft.

The cam-curve may be constructed on paper by means of tangents, as shown in Fig. 168. If *c* represents the centre of the cam-shaft, and

c r the distance from the centre of the cam-shaft to the centre of the stamp-stem, the circle described about c , with c r as a radius, is the developing circle of the involute. The distance, representing the height to which the stamp is to be lifted, is laid off upon the circumference of this circle, as from the point 1, which distance is subdivided into a convenient number of equal parts, determining, as in Fig. 168, the points

FIG. 168.



CAM, OR "WIPER"

2, 3, 4, to 13. From each one of these points in the circle, a tangent is drawn, on which is laid off a distance equal to the length of arc between the point 1 and the point from which the tangent is drawn. All the points thus determined in the tangent lines are points in the cam-curve, and may be connected as shown in the figure, thus producing the line for the face of the cam.

In practice, the line of curvature is produced by cutting from a thin board a circular piece, the radius of which is equal to the horizontal distance from the centre of the cam-shaft to the centre of the stamp-stem. At a given point on the periphery of the circular piece is fixed one end of a thread, which must have the length of the greatest desired lift of the stamp, and to the other end of which is attached a pencil-point. The circular piece, with the attached thread wound on the periphery of the circle, is laid on a smooth board, on which the line is to be traced, and the thread being constantly stretched to its farthest reach, is unwound until it forms a tangent to the circle at the point where the other end is attached. The line described by the pencil-point is the desired curve.

Some builders slightly modify this curve, giving to the cam-arm a greater curvature near each of its ends, in order that the cam in its revolution may come into contact with the tappet at the least practicable distance from the cam-shaft, where the concussion is less than at a greater distance, and to diminish the friction between the extreme end of the cam and the face of the tappet. The face of the cam is 2 or $2\frac{1}{2}$ in. wide. Its extreme end is fashioned so as to correspond to the outer edge of the tappet, which is circular. The cam is placed as near the stamp-stem as practicable without coming into contact with it. The cams are caused to revolve by means of the cam-shaft, to which they are secured by one or sometimes two keys or wedges.

Cam-shaft.—The cam-shaft is a round shaft of iron, which is smoothly turned and finished, having one or two key-seats or grooves cut in it lengthwise, for the purpose of securing the cams in their places. The shaft rests in boxes, which are usually supported by shoulders cut on the upright posts of the battery-frame. Cam-shafts vary in diameter from 4 to 6 or 7 in., according to the number of cams to be fixed upon them, and the weight of the stamps to be raised. In some mills, a single cam-shaft is made long enough to carry all the cams for as many batteries as there may be. In Nevada and California, however, short cam-shafts are in general use, a separate shaft being employed for each battery, or, in many cases, one shaft for two batteries. Separate cam-shafts are preferred, on account of the independence of each battery, so that if one be stopped by any accident to the cams or stamps, or for repairs of any kind, the operation of the others is uninterrupted. Each shaft in such case is driven by its proper pulley, which receives its motion by means of belting from a counter-shaft. The order in which the stamps are to fall being determined, it is carried into effect by fixing the cams on the shaft in such position that each cam, by the revolution of the shaft, will lift its respective stamp at the desired moment. For this purpose, the key-seats cut in the hub of the cam must be determined with care; one common key-seat being cut on the cam-shaft, when the desired position of any given cam has been ascertained, the key-seat in the hub is cut to correspond with that of the shaft.

Props.—When it becomes necessary to hang up a stamp so that the cam may revolve without reaching the tappet, it is supported by a prop, stud, or "finger" *x* (fig. 156). The lower end of the stud, of which there is one for each stamp, is pivoted on a small shaft fixed across the battery from end to end, resting in boxes, which are secured to the uprights. Each stud is just long enough to support the stamp, when placed under the tappet, at a height which is about 1 in. above the highest lift given by the cam. To bring the end of the stud into this position when desired, the workman lays a smooth stick on the face of

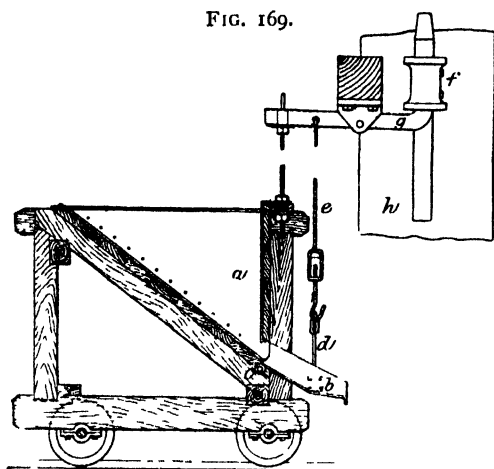
the cam as it is rising to the tappet, and holds it there while the stamp is lifted. The stick is as wide as the face of the cam, and long enough to be held conveniently, and $1\frac{1}{2}$ in. thick at the end which comes between the cam and tappet. By this means the stamp is raised high enough for the stud to be put in place, which being done, the stamp is supported above the reach of the cam. To set it again in motion, the operation is repeated, the stud being withdrawn at the moment when the stick on the face of the cam has lifted the stamp clear of its support. To lift the stamp-stem, a chain or rope pulley is provided above the battery, and should be arranged to travel on rails so that it can be readily shifted over any particular stamp.

Feeding.—Much of the effectiveness of the stamps depends on the degree of care devoted to keeping the working parts in good condition, and on the regularity with which they are supplied with ore. This is commonly done by hand labour, the rock being shovelled in at such a rate as it is crushed and discharged. In some mills, however, automatic feeders are employed, which give satisfaction. These consist of a hopper filled with ore, from which a trough or chute leads to the feed-opening of the battery, so inclined that the ore will slide down from the hopper to the battery if the chute, which is hung on a pivot, be agitated. A rod is attached to the chute, and so placed that the tappet of the stamp, when the latter gets so low as to require an additional supply of rock, will strike its upper end, thus giving a shock which causes the ore to move down and fall into the battery. But a great objection to most automatic feeders is that they are arranged to supply one constant quantity, and make no difference in the work for the several parts of the mortar under varying conditions. With hand-feeding, the ore is received into a large bin or pocket, the floor of which is made in such a way that the ore will run easily towards the mortar. Here, in a space of 6 to 12 ft., stand the feeders, whose duty consists in keeping a constant amount (generally 2 in. deep) of ore between the shoes and dies, whereby the drop of the stamps is maintained at an equable height.

Automatic ore-feeders are broadly of one type, as just described. Stanford's feeder, as made by Joshua Hendy, 49, Fremont Street, San Francisco, is shown in Fig. 169. It consists of a hopper *a*, with adjustable spout *b*, swung on trunnions *c*, and attached to a cross-bar *d*, suspended from an adjustable rod *e*. A feeding-tappet *f* is keyed upon the battery-post *h*, and a lever *g* rotating on pivots is fixed to the rod *e*, so as to be struck by the cam-tappet, the lever *g* being forked that it may span the stem. While the battery is supplied with sufficient ore, the tappet does not descend far enough to strike the end of the feeding-rod; when the ore gets low, the tappet does strike the rod, and the effect is an oscillation of the front spout on its trunnions, whereby the ore is thrown forward.

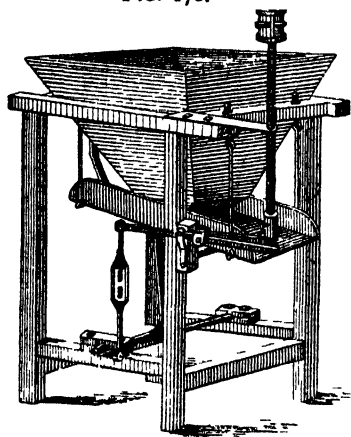
The apparatus being on wheels can be readily removed. It is simple in construction and operation, maintains the feed at any desired degree, and materially reduces the wear and tear of the stamps, while increasing their duty, it is said, 25 per cent. It seems best adapted for dry-crushing.

FIG. 169.



STANFORD'S ORE-FEEDER.

FIG. 170.



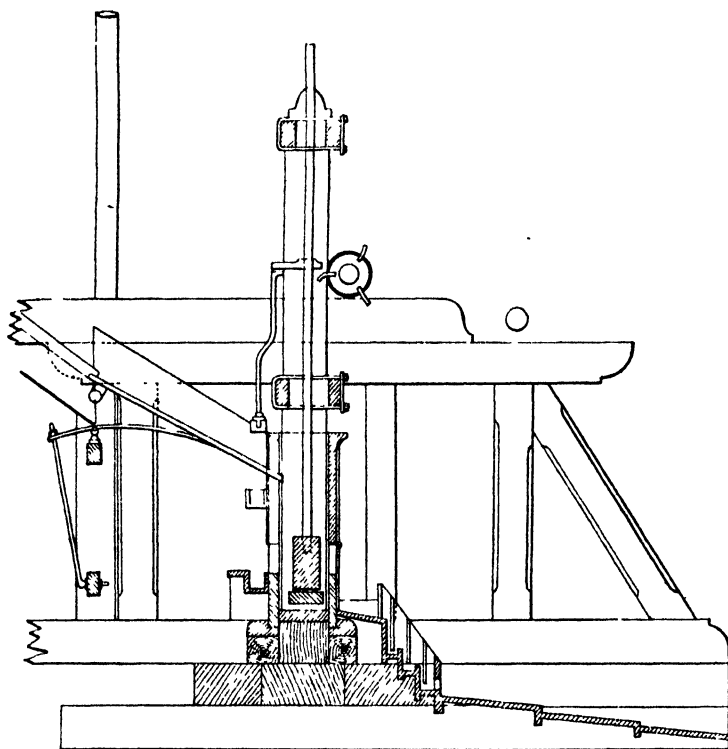
TULLOCH'S ORE-FEEDER.

The Tulloch feeder, also made by Hendy, is likewise largely used. It is illustrated in Fig. 170. In Fig. 171 is shown the self-feeding hopper employed by the Port Phillip Gold Mining Co.; and in Fig. 172, the roller ore-feeder made by the Golden State and Miners Ironworks, San Francisco.

Hendy's "Challenge" feeder is perhaps the most popular for wet, sticky ores. The mode of attaching it (to the second stamp in the battery) is shown in Fig. 173: *a*, tappet; *b*, lever; *c*, lower guide; *d*, hopper; *e*, carrier-table; *f*, shoot; *g*, bumper; *h*, stamp-stem. This feeder is said to effect a reduction of the wear and tear by 15 per cent., and an increase of duty by 20 per cent. It is by many considered the best feeder for all ores, but is more costly than some other forms. Recently an improvement has been introduced, whereby the feed side of the mortar is rendered more accessible.

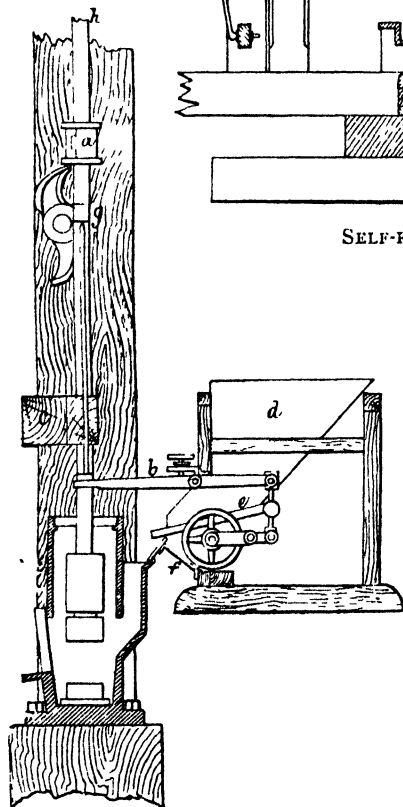
Recently attempts have been made to introduce automatic feeders for supplying mercury to the mortars of batteries where the practice of having mercury in the mortar prevails. Dubois' apparatus, made by Whitney & Marshall, 22, Fremont Street, San Francisco, seems to be the only one in the market. There are obviously many advantages to be derived from the automatic feeding of mercury, that is to say, if the practice of putting mercury into the mortars deserves to be supported at all.

FIG. 171.



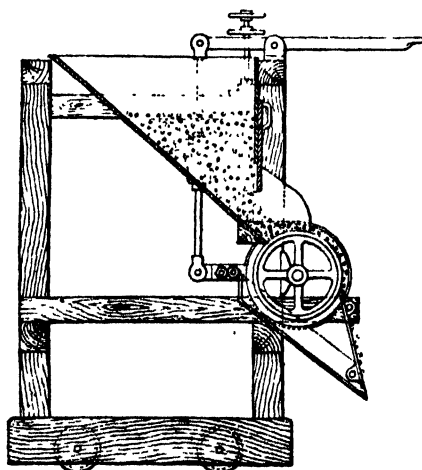
SELF-FOLDING HOPPER.

FIG. 173.



HENDY'S ORE-FEEDER.

FIG. 172.



ROLLER ORE-FEEDER.

Water.—The quantity of water used in wet crushing depends in a great measure upon the character of the ore and the degree of fineness to which it is crushed. In Victoria, the proportion varies from 30 to 1200 gal. per stamp-head per hour, though 300 to 500 gal. would appear to satisfy all requirements. In America, about 93 gal. is commonly thought sufficient. Egleston says the consumption in Nevada and California is 200 to 300 cub. ft.* per ton of rock stamped, or $\frac{1}{3}$ to $\frac{1}{2}$ cub. ft. per stamp per minute, for all purposes, leaving probably about $\frac{1}{4}$ cub. ft. for the batteries alone (or nearly 94 gal. per hour). In Colorado, the amount is 28 cub. ft. per ton of rich ore, and 33 cub. ft. per ton of poor ore. The cub. ft. of ore averaging 108 to 125 lb., this will give $\frac{1}{4}$ cub. ft. per stamp per minute, or about the same as Nevada and California. When the water used is purchased from a "ditch company," it is measured by the miners' "inch" (see p. 306).

The water is fed to the stamps by horizontal piping just above the feed-slot of the mortar, with orifices opposite each stamp, capable of being closed if necessary; or the main feed may be brought higher up, and vertical supplies be carried from it down to each stamp, with valves at the ends. The main is often a 3-in. gas-pipe. A second main of half the size is placed in front, to help carry off the pulp. In America, arrangements are always made to warm the water supplied to the stamps, by waste steam or otherwise, during winter.

Tables of Dimensions and Duty.—The annexed tables reveal at a glance the dimensions and working results of a number of mills in various parts of the world, including some that may be considered representative.

Mill or District.	Weight of stamps.	Drops per minute.	Depth of Drop.	Crushed per 24 hours.	No. of stamps.	Duty per stamp.	H.P. per stamp.	Holes per sq. in. of grating.	Water per stamp per 24 hours.	Mercury used per stamp.	Mercury lost per stamp.
	lb.		in	tons.		tons.			gal.	lb.	oz.
Gras Valley	850	61	10	40	20	2	..	..	..	..	..
"	700	68	10	32	20	1'12	..	..	..	..	..
Eureka ..	950	80	9	..	60	2½-3	..	..	..	..	..
Brunswick ..	..	..	..	160	56	3	..	..	..	..	..
Keystone ..	750	75-80	..	75-80	40	2	..	..	..	..	..
Idaho ..	950	80	9	..	..	..	..	..	..	..	..
Metacom ..	900	90	10	..	..	..	..	..	..	..	..
Port Phillip	672	75	..	..	56	2'2	1	..	..	..	..
"	896	75	..	..	24	3	1½	..	..	..	..
Nova Scotia	650 cwt.	55	6-9	..	..	1-1½	..	..	..	..	..
Ballarat ..	4-8½	50-85	7-10	..	..	1-4	1-2	40-200	950-8640	5-75	1-8
Beechworth	4½-7½	40-90	5-14	..	..	½-4	½-1½	60-140	720-11,520	5-70	½-8
Sandhurst ..	5-8	25-75	6-18	..	..	1-3½	½-2	64-140	4000-8640	10-40	½-5½
Maryborough	4½-8	50-75	6-22	..	..	1-3	½-2½	70-144	900-8640	3-30	1½-8
Castlemaine	4½-8	35-75	6-15	..	..	1-3½	½-2	40-144	4800-12,960	6-40	½-24
Ararat ..	5-6½	60-72	7½-10	..	..	1½-1½	½	90-120	4320-12,960	6-47	½-7
Gippsland ..	6-7½	60-80	7-10	..	..	1½-2	¾-1½	70-250	1600-25,000	10-37	½-32

gal. = 277½ cub. in.; 1728 cub. in. = 1 cub. ft.

Name of Mill.	Length of stem.	Diam. of stem.	Weight of stem.	Height of shoe.	Diam. of shoe.	Weight of shoe.	Height of boss.	Diam. of boss.	Height of tappet.	Diam. of tappet.	Weight of tappet.	Height of die.	Diam. of die.	Weight of die.
	ft.	in.	lb.	in.	in.	lb.	in.	in.	in.	in.	lb.	in.	in.	lb.
Douglass	12 $\frac{1}{2}$	2 $\frac{7}{8}$	290	9	8	115	18	8	12	8	120	..	..	..
Cons. Virginia ..	13	3 $\frac{1}{4}$	320	7	8	110	16	8	10	7 $\frac{1}{2}$	95	..	..	..
Lincoln	13	3 $\frac{3}{4}$	320	7	8 $\frac{1}{2}$	119	18	8 $\frac{1}{2}$	10	7 $\frac{1}{2}$	93	5 $\frac{1}{2}$	8 $\frac{1}{2}$	99
Brunswick	15	3 $\frac{1}{2}$	375	10	9	125	18	8	12	9	125	..	..	..
Electric	11 $\frac{3}{4}$	3	258	8	8 $\frac{1}{8}$	123	16	8 $\frac{1}{2}$	8 $\frac{1}{2}$	7 $\frac{1}{2}$	83	6	8 $\frac{1}{8}$	100
Eureka	14	3 $\frac{1}{2}$	450	..	..	160	..	..	..	..	120	..	..	120
Keystone	..	..	..	..	..	100	..	..	..	..	100	..	..	113
Stanford	..	..	..	..	..	120	..	..	..	..	114	..	..	..
Walhalla	10 $\frac{1}{4}$	3 $\frac{1}{4}$	..	9	10	..	14	10	..	..	..	..	..	..

The wide differences in the figures given in the preceding tables are a sufficient warning that the weight and speed of stamps most suitable to a particular ore, can only be ascertained by actual trial of that ore. It is vain to expect that the best attainable working results can be got by applying the figures found successful in one case to another case apparently similar. Ores vary in the most marked degree, especially in resistance to disintegration, and such variances cannot be judged in an out-of-hand fashion. Therefore careful experiment with large and representative samples of the ore should precede the choice of dimensions for the stamp battery.

In general terms, it may be said that there is a tendency towards the adoption of heavy stamps, with slight fall and rapid stroke, in preference to the opposite pattern which ruled at one time. A series of experiments made at Metacom in America, clearly showed that the effective capacity of the battery was very largely increased by augmenting the speed of the stamps. In a dry stamping silver mill the increase of work was 244 per cent. for an increase of speed amounting to 85 per cent. ; or in other words, by quickening from 60 to 102 drops a minute the outturn rose from less than 1 to over 3 tons per stamp. These figures will not be attained in wet crushing gold ore, because in dry crushing it is the compression of air by the fall of the stamp which alone forces the pulp out of the mortar, while in wet crushing the movement of the water greatly assists in freeing the battery, and it must be borne in mind that the rate of escape from the battery is the rate of effective work done by the stamps.

Against the extra horse-power required to move the heavier stamp, there must be set the gain in efficiency, which is always in advance. There is also a saving in labour. But certain considerations control the limits to which these features may be economically developed. For instance, excessive weight or speed would soon injure the structure of the battery, and excessive speed alone would be abortive, as it would not allow the stamp to fall at all, the rapidly revolving cam keeping it suspended.

Life of Battery.—The duration of the several parts of a stamp battery is very unequal. An ordinary die should last about twice as long as the shoe, or say six weeks to two months; the rate of wear is commonly $\frac{1}{2}$ lb. to 1 lb. per ton of ore treated. The conditions which govern the rate of wear are the speed of the battery, regularity of feeding, hardness of ore, and quality of iron. Good cast iron is as enduring as steel, but much of the cast iron employed for this purpose in remote localities is far from good. Shoes generally last four to six weeks, wearing at the rate of 1 to $1\frac{1}{2}$ lb. per ton of ore put through.

Stamp stems, especially when running in wooden guides, should last three or four years, and, with regular feeding, very much longer; the jar occasioned by a short supply of ore in the mortar is apt to cause them to break off at the ends. When this happens, the injury is remedied by welding on a piece, which may be done repeatedly. Stamp heads sometimes split when not bound with iron rings, and the sockets become enlarged by wear; but the latter drawback may be overcome by inserting wooden or iron wedges. Tappets should endure 4 or 5 years, but are sometimes split through wedging up too tightly.

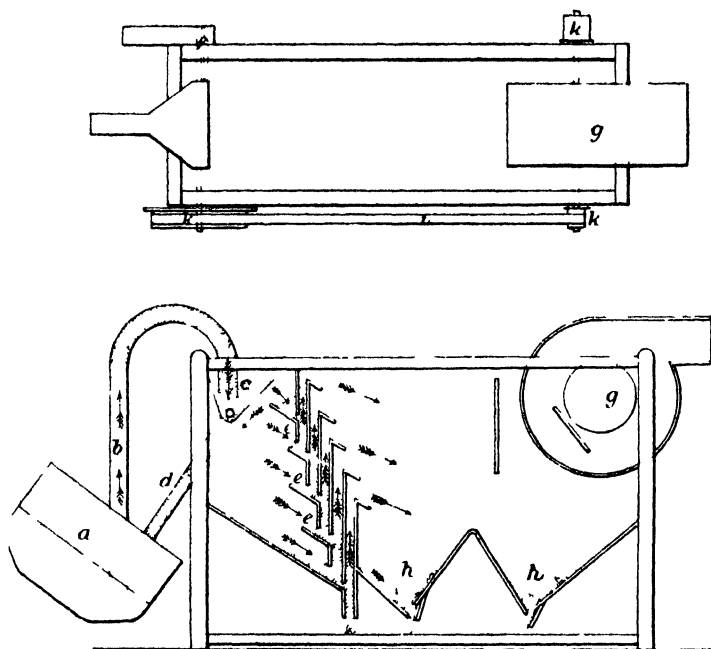
In order to change a stamp-stem, an operation occupying about half an hour, the whole battery must be hung up on the props already described (p. 441). The die and shoe are placed on the foot-plate, and three or four strips of wood, $1\frac{1}{2}$ in. wide, are put into the stem socket exactly as if the battery were new. The stem is then let down to within about 3 in. of the die, and the tappet is keyed on in this position; the stem and tappet are then hung up upon the prop. While the whole of the rest of the battery remains hung up, a strip of wood $2\frac{1}{2}$ ft. long and 2 in. wide, covered with leather, is placed between the cam and the tappet and held in one hand, while with the other the workman seizes the handle of the prop and pulls it out from under the tappet. The stem falls into the socket of the die. Keeping the stick in its place, the workman allows this single stem to work until it is quite fast in the die; he then hangs it up again and going below raises the shoe to its proper level, supporting it there, and then sets the tappet in its proper place. This is a delicate operation requiring considerable skill, as for a time the stem is held up by the pulley alone, and just the right moment must be chosen to set the stick under the cam. If the stick is not held exactly in the proper position, it may result in breaking the stem.

Dry Stamping.—The idea of stamping the ore dry is by no means new; in fact it was probably the original method, but gave way to wet stamping because it was found that water assisted in clearing the mortar of fine matters, and served as an aid to separation of the valuable and valueless matters afterwards. Whether it might not be better to stamp dry and not add the water till the ore is in a fine state is, however,

exercising the minds of some mill managers, since means have been perfected for drawing away the fine particles from the mortar as soon as they attain the desired degree of comminution.

An obvious method for withdrawing the powdered ore is by means of an aspirator, as shown in Fig. 174, which almost explains itself. The aspirator is there shown fitted to Fisher's revolving bed stamp.

FIG. 174.



ASPIRATOR FOR USE IN DRY STAMPING.

as the matters in the pan *a* are sufficiently fine to ascend with the air current, they pass up the tube *b*; and on reaching the hopper *c*, an opportunity is provided, by a worm *f*, for all the grosser particles to fall back by *d* into the pan *a* for further reduction. Those particles which still float in the air current pass through a series of apertures *e*, and gradually find their way to the floor, which is arranged in the form of a succession of hoppers *h*, each delivering a finer sized grade of mineral than the one preceding it, as the end where the aspirator *g* is situated is approached. The mechanism is driven by belts and pulleys *i k*.

Recently, W. H. Nash, of Sydney, has brought under the notice of the Victorian Mining Department a dry-stamping process invented by him for extracting gold from quartz. The apparatus consists of a tall chimney of sheet-iron or other suitable material attached to an air-tight enclosure

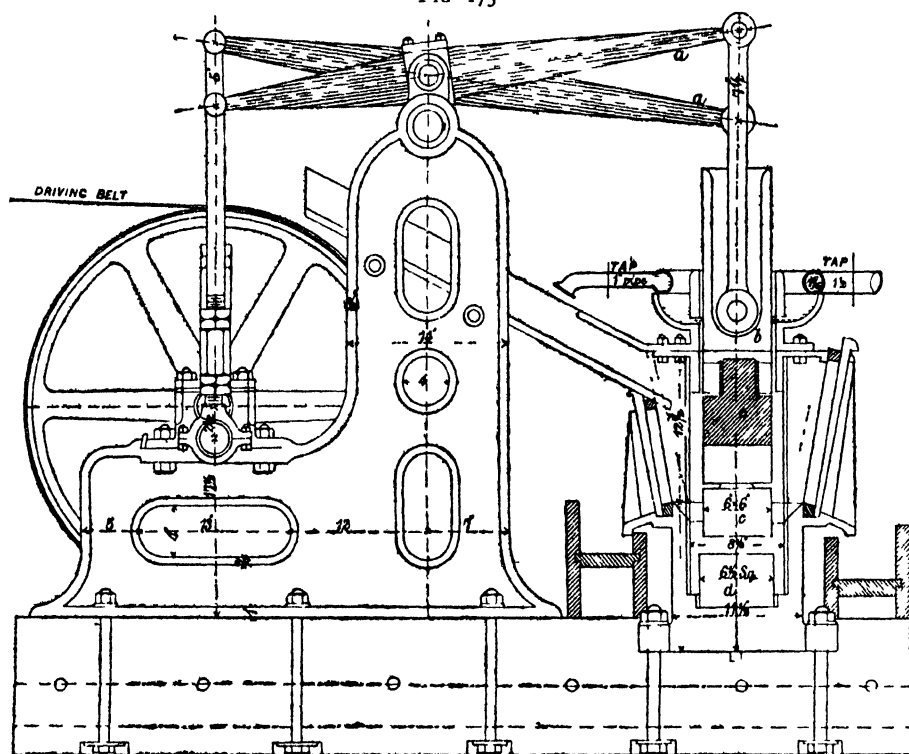
in front of the gratings of the stamping-boxes, and communicating with the external air through the gratings and stamping-boxes only. Water not being used, the stamps, stamping-boxes, and quartz quickly become warm in working, and the heat thus generated causes a strong draught of warm air to pass up the chimney, which, being supplied only through the gratings, draws the fine powder with it. This arrangement, it is said, has been found in practice to discharge the stamping-boxes as rapidly and effectually as if water were used. Combined with this apparatus is an enclosed elevator, into which the pulverised quartz falls as soon as it has passed through the gratings, and which, continually working, raises it to a high level for subsequent treatment. Nash claims that the process gives a much higher average yield of gold per ton, and that the cost of treatment is less, there being no handling. The particular process invented by Nash has not yet been introduced on the Victorian gold-fields; but some years ago a machine for dry-stamping quartz was erected at Ballarat by the late Mr. Costin, and it pulverised finer than an ordinary battery, at a cost of only $6\frac{1}{2}d.$ per ton.

Special Forms of Stamp.—Within recent years, several forms of stamp have been introduced which exhibit a departure from the ordinary rules governing the construction of batteries. They are based mainly upon one principle, which is to have at most 2 stamps in a battery, working at great speed (150 to 200 blows a minute), and weighing only 2 to 4 cwt. each. They thus present several advantages, among which may be specially mentioned—(1) their lightness and consequent portability, (2) the rapid discharge effected by the all-round delivery, and (3) the sharpness of the blow. The principal forms will be described in alphabetic order.

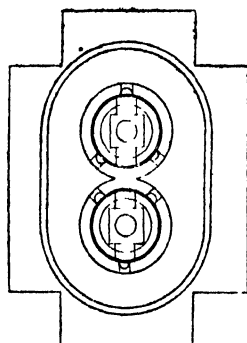
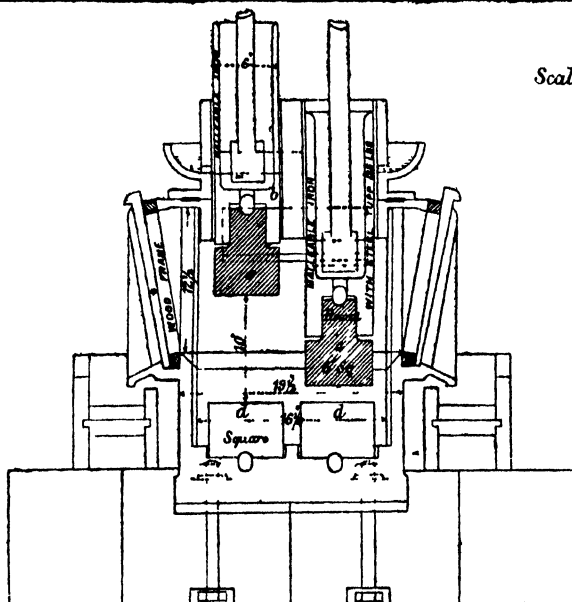
Dunham's.—The chief feature of this stamper consists in the employment of 2 plate-springs, balancing each other, mounted on bearings placed in rear of the centre of their length, and driven at the short ends by connecting-rods (whose length is adjusted to the wear of the shoes), driving-shaft, and pulley, in the usual manner. The machine is shown in Fig. 175, being of a size to crush 12 to 15 tons per 24 hours, at 250 blows a minute. Attached to the long arms of the springs *a* are suspended cast-steel tupps *b*, with forged and tempered steel shoes *c*, striking elastic blows on the material lying on the dies *d*. With some qualities of material, each head can deliver 300 blows a minute. A point in which this stamp excels all others is that the gratings are on each of the 4 sides of the battery. The total weight of the machine is $1\frac{3}{8}$ ton, and the heaviest part is less than $3\frac{1}{2}$ cwt. The same patentee has a much more portable machine for prospecting purposes, in which the heaviest part is less than 100 lb.

Fisher's.—Fisher's patent rotating-bed stamping mill differs essen-

FIG 175

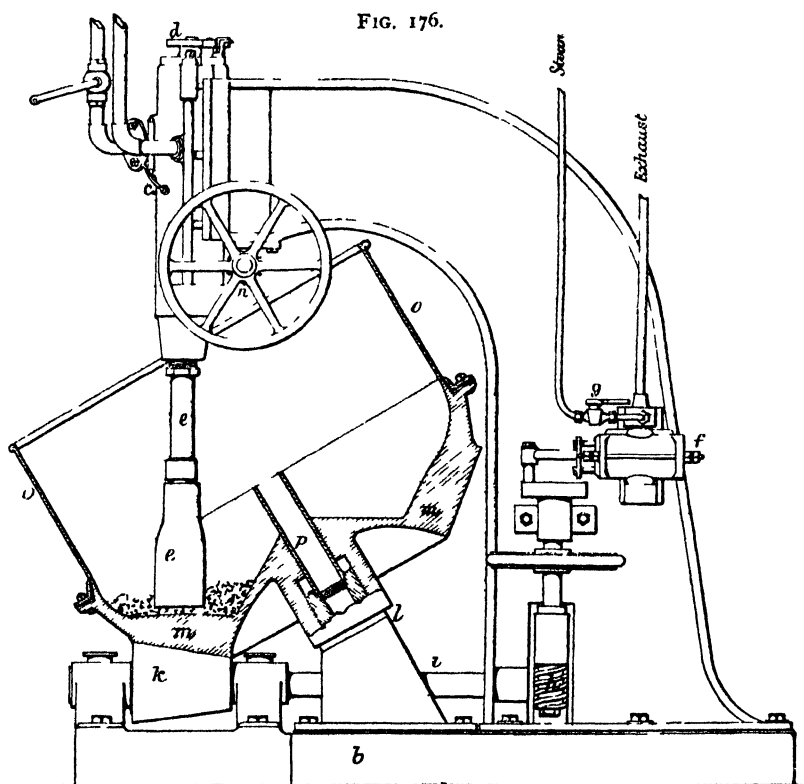


Scale $\frac{1}{4}$ in = 1 ft



DUNHAM'S RECOIL STAMP

tially from the other forms dealt with in this section, and is shown in Fig. 176. The working parts are as follows: A strong cast-iron frame is bolted to a substantial bed-plate *b*, and carrying at its upper end the cylinder *c*; in this is fitted a piston, which, by admitting steam or compressed air through the supply-pipe, as shown, is made to reciprocate up and down at a very high velocity, namely (according to the pressure used) from 400 to 800 strokes per minute. On the top of the piston is fixed a nut working on a rifled or twisted bar, which passes through the upper end of the cylinder, and is prolonged sufficiently far through the cover to admit of a ratchet-wheel, shown at *d*, being keyed to it, into which a pall is fitted in such a manner as to allow of the stamp-head *e* revolving on its descent only, viz. when coming into contact with the



FISHER'S ROTATING-BED STAMP.

material under treatment. The piston and piston-rod are formed out of one solid piece of steel, and the stamp-head is also of steel, with the exception of that portion that actually strikes the material, which is of

chilled iron. It is very firmly secured to the piston-rod, and can be disconnected and replaced in a few minutes at a trifling cost. At *f* is a small horizontal oscillating cylinder, which is set in motion by opening the cock *g*, and supplying steam or air, as the case may be, through the supply-pipe. The piston-rod being attached, as shown, to a small disc-crank, causes the worm-wheel *h* to revolve, giving motion to the shaft *i* and friction-roller *k*, which latter, having a considerable bearing surface, induces by friction the rotation on its bearing *l* of the large rotating bed or anvil *m*, which is shown in section. It will be noticed that the part of the bed immediately under the stamp-head is made of great strength and solidity, the entire pan weighing about 15 cwt. At *n* is a hand-wheel, which enables the attendant to raise or lower the cylinder whilst the machine is in motion, thus diminishing or increasing the distance between the bottom of the stamp-head and the inner surface of the rotating bed, as may be desired, from $\frac{1}{8}$ to $4\frac{1}{2}$ in. The stamp-head not being allowed to strike the anvil, it has simply to overcome the resistance due to the crushing. The material to be crushed, having been first reduced to pieces of a manageable size, is fed into the vessel *m* whilst the stamp is working, and can be reduced by the latter in a very short time to any degree of fineness required, the reduction being greatly assisted by the three combined motions, viz.: of the revolution of the bed *m* (which ensures a constant charge and fresh supply of material at every stroke), the descending stamp, and the revolving and consequently grinding motion of the same at the period of contact, which motions, being concentrated at the same moment on a certain area of material, are said to have great effect in crushing, grinding, and pulverising, either with or without water.

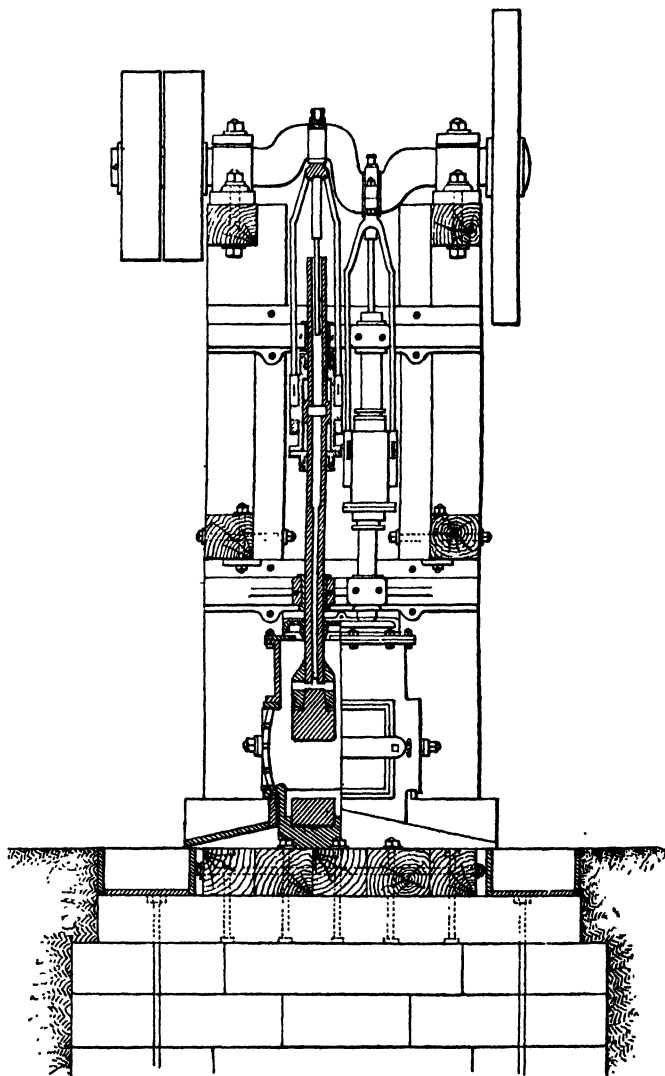
The material thus rapidly and effectually treated may be, in the case of quartz, discharged in the usual way, through the grating or wire gauze surrounding the bed, as shown at *o*, which, of course, can be of any form or mesh required; or, in the event of its being used as a flash battery, could be delivered through the pipe *p*, which can be made adjustable. It need scarcely be added that mercury for amalgamation or other ordinary means for collecting gold contained in quartz, may be applied in the usual way.

Hope's.—A very recent suggestion is the compound battery of A. Hope, Prahnan, which consists of two or more rows of stamps placed in one box and driven at equal or unequal speeds. Where round stamps are used, the back row, or preparatory crushers, are made to rotate in an opposite direction to the front row, or crushers proper, which have a greater crushing surface than the back stamps. It is claimed for this system that the cost of erection and working is reduced, and the efficiency increased. It seems quite probable that the latter

claim is sound, as all four sides of the battery can carry screens, and thus multiply the area of outlet.

Husband's.—This pneumatic stamp (Fig. 177), was originally devised as an improvement on the old-fashioned stamps used in Cornwall for reducing tin ore ; but its success in that industry has led to its adoption

FIG. 177.

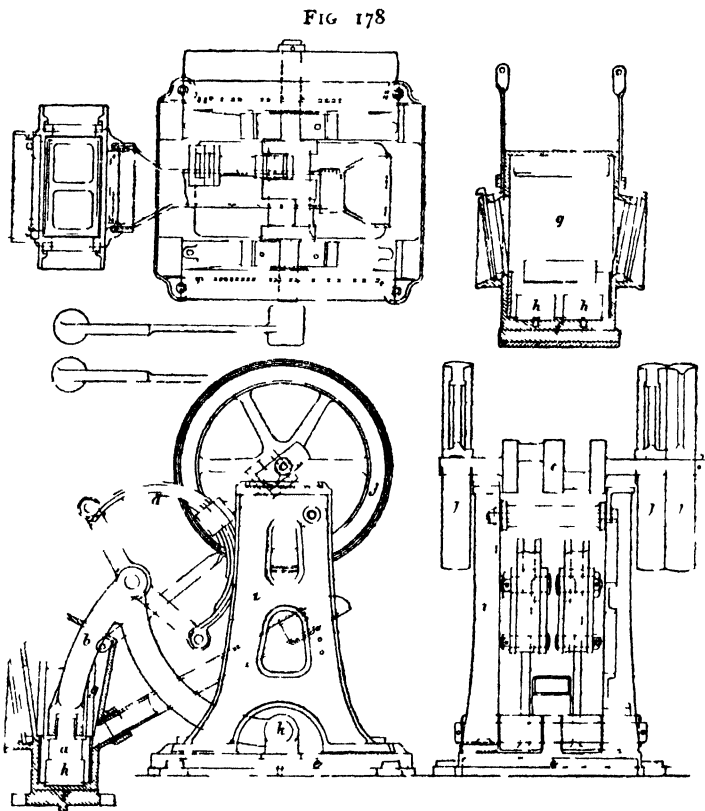


HUSBAND'S STAMP.

also in gold mining. Its portability is an important feature. It can be transported anywhere and erected in a few hours, requiring no heavy

outlay for foundations. In all parts of the machine, wrought iron replaces cast-iron as much as possible, thus reducing at once risk of breakage, weight, and consequent cost for transport. When striking 140 to 145 blows a minute, the output is said to be 7 to 10 tons per head per 24 hours, varying with the hardness of the ore. The head and lifter weigh less than 3 cwt, and the whole machine does not exceed 4 tons. The power required is 5 to 6 horse-power per head

Patterson's.—In Fig. 178 is shown the "elephant" stamp invented by John Patterson, which in appearance and character differs widely from other types of stamp heretofore used, and is stated to suit equally well for either wet or dry stamping. The most striking features in this



PATTERSON'S "ELEPHANT."

stamp are the absence of all guide-rods and brakes, and extreme compactness and fewness of working parts, which reduce the friction and consequent wear. The two stamp-heads *a* are fixed in the ordinary way into two hammered scrap-iron levers *b*, which are attached to a 2-throw crank-shaft *c* by the intervention of two powerful semi-circular steel springs *d*. A great economy of power is effected by the peculiar action

of the springs, storing up the force of the recoil from the blows of the stamp, which force is given out on the next return of the cranks. The power is further conserved by each stamp balancing its companion.

Some of the advantages said to be possessed by these stamps are that they are portable, can be taken to pieces and re-erected in a few hours, and need no special building. Foundations for the bed-plates *e* as also for the anvil-block *f* under the coffer or mortar *g*, may consist of stone, concrete, or wood. The stamps can be fixed in any situation where there is a driving-shaft or wheel, and are independent of the position or nature of the motive power. All the parts are so strong as to resist any unusual strain. As the shoes or heads *a* wear, no alteration is needed, as the springs and flexible connections allow for varying distances between the face of the stamp-head *a* and anvil *h*, whether due to the wear of these parts or to the quantity of stuff on the anvil.

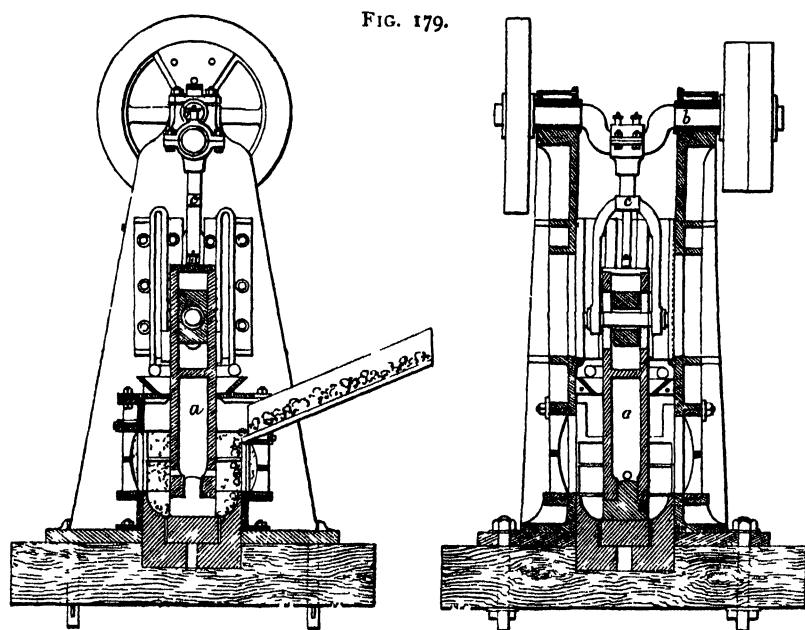
The quick napping nature of the blows ensures that there is no sliming of the stuff. Each head delivers 130 to 140 blows per minute. By varying the speed, the effect of the blows can be regulated at will to suit the degree of hardness or toughness of the stuff operated upon. The two heads will stamp 20 to 24 tons per diem of quartz, to the ordinary fineness. About 5 to 10 H.P. indicated will drive the stamps, according to the speed required. The continuous flow of the stamped stuff, owing to the quick action of these stamps, enables a much greater quantity to pass in a given time through the screens: the screen area, however, is about double what can be obtained in ordinary batteries per stamp-head. Where stamping and amalgamation are carried on simultaneously, the copper plates inside and outside the mortar require to present an area for deposit proportional to the work being got through by these stamps, so that the continuously discharged stuff may have ample time to be distributed over the plates.

The makers enumerate the following points as needing attention in erecting and working these stamps. The foundation, as in all other stamps, must be solid and unyielding. The die must possess sufficient inertia to withstand the powerful blows of the stamps, and to secure the maximum effect of the blows on the material operated upon, and not let any of it be wasted on the surrounding ground.

After the foundations are prepared, the top is levelled, and the holding-down bolts are in their places, the base-plate *e* is put into position, levelled accurately, and fixed with the holding-down bolts; the side-frames *i* are erected with their stay-pipe and through-bolt, and securely bolted to the base-plate already fixed; pedestals are put on top of side frames, and the double-crank shaft *c* is dropped in and the pulleys *j* strung on, seeing that they are clean and well oiled; the mortar-box *g* is let loosely into its place; the stamp-levers *b* are put in position, and the fulcrum-pin *k* and loose collar passed

The "elephant" stamp of slightly different pattern tried on the Pacific coast of America did not give altogether satisfactory results ; but the manufacturers now claim to have overcome the difficulties.

Sholl's.—Sholl's pneumatic stamp is shown in Fig. 179 : *a*, pneumatic



SHOLL'S PNEUMATIC STAMP.

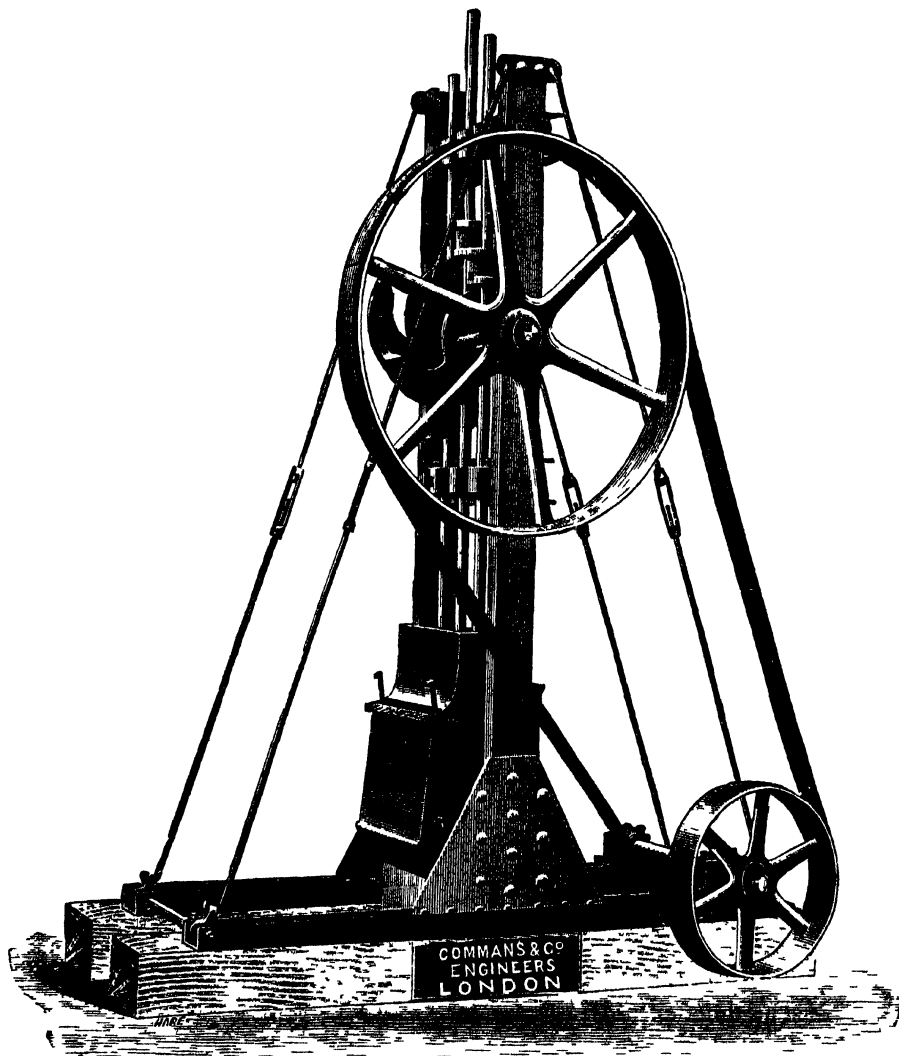
cylinder and ram ; *b*, crank-shaft ; *c*, forked connecting-rod. The principal feature is the absence of glands, stuffing-boxes, and frictional surfaces, into which the pulverised ore can be flashed. The weight of the blow is varied by the speed. Great economy of power is obtained by storing compressed air in the cylinders, which is again utilised by expansion on the return of the crank, causing an elastic spring of air, released at every stroke. Ore which will pass through a 6-in. ring can be fed in. The shoe will bear the loss of 2 cwt. of its face before needing adjustment. A single head, to crush 12 to 15 tons per 24 hours, costs about 210*l.* ; a double head, to crush 20 to 22 tons, 250*l.* The heaviest piece need not exceed 3 cwt.

Prospecting Stamps.—Stamp batteries constructed for prospecting, differ from ordinary batteries chiefly in being of light build, so as to be readily removable as circumstances may require. They need no permanent foundation, and can be driven by manual power.

Fig. 180 represents a neat and substantial form of portable battery specially designed by Commans & Co., 52, Gracechurch Street, London,

for carrying out prospecting work in new fields. It consists of 3 heads of stamps of a weight of about 100 lb. each. All the wearing parts are of steel, and the battery is so designed that it can be readily taken apart and transported from place to place on mule back or in ox wagon. The

FIG 180



PROSPECTING STAMPS.

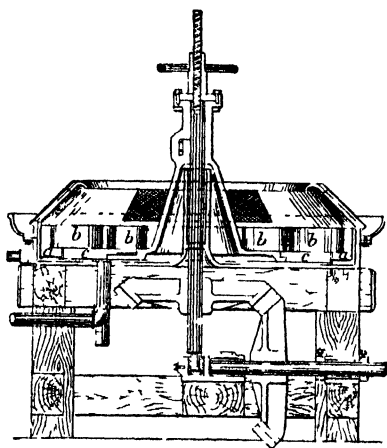
heaviest piece is the mortar box, which weighs about 3 cwt., the total weight of the complete battery being 1 ton. Though so light and portable, it is capable of crushing 1 to 2 cwt. per hour of any ordinary

quartz, when running at 60 drops a minute. The battery is arranged to be driven either by hand or by power. In the latter case a simple form of horse or bullock gear is usually employed, connected direct by a universal coupling to the end of the counter shaft. Amalgamating tables are fixed in front of the mortar box, as in larger batteries, for catching the gold.

Pulverisers.—A class of machines, designed to avoid some of the drawbacks of stamping batteries, has lately been introduced under the general term of pulverisers. Some work on the principle of grinding or tritulating the material to a finely pulverulent condition, without the aid of blows while others rely upon repeated concussion. Their chief forms are as follows :

Howland's.—The Howland pulveriser is shown in Fig. 181. *a b c* are the wearing parts, which are made of the hardest iron, in duplicate, and can be changed with less trouble than the shoes of the stamp-mill, no keys or bolts being required to hold them. *a* is an L-shaped ring of the full diameter of the inside of the machine. *b* are rings or rolls 10 in. in diameter, $4\frac{1}{2}$ in. face ; 12 of these are used upon the disc-plate *c*, which must run at 175 to 180 revolutions per minute ; the rings *b* rest partly upon the outer ring *a*, which is stationary. The motion of the disc-plate *c* imparts a peculiar motion to the rings or rolls *b*, they turning as it were upon a central axis, spinning like tops, while the whole 12 at the same time run around the large circle, being thrown off by centrifugal force. The pulverising is done between the rolls or rings *b* and the outer ring *a*. The aggregate weight of all the wearing parts is about 2135 lb. The machine will pulverise, it is said, 500 to 750 tons of ore to 40-mesh and finer, with one set of the wearing parts, which is less wear or loss of iron than in stamping the same amount of ore to same grade of fineness. The ore is fed through the opening in the top of the bonnet-casting ; immediately on falling upon the revolving disc-plate, it is carried outward by centrifugal force to the rings or rolls, and when pulverised fine enough, is ejected through the screens to a circular trough conveying it to copper plates for amalgamation, or run into tanks for settling. The machine can also be constructed for dry pulver-

FIG. 181.



HOWLAND'S PULVERISER.

ising, when it will accomplish about one-fourth less work. The frame for the machine is made of Southern pine timber, mortised and tenoned throughout, and held by strong joint-bolts. An automatic feeder is provided especially for this machine, that will feed the ore continuously, as well as a rock-breaker, adapted to breaking the ore to the proper size for the machine, which must be no larger than 1 in. This machine is said to pulverise wet, hard quartz rock to a fineness that will pass through a 40-mesh screen, at the rate of a ton per hour, and will pulverise dry, to pass through a 60-mesh screen, $\frac{1}{2}$ to $\frac{3}{4}$ ton per hour. Every exposed part of what might be termed the wearing castings, namely, the pan-ring, the revolving disc-plate, and the rolls, are continually passing and re-passing each other at different points with a rolling and grinding action, ensuring a maximum of wear from a minimum of metal. These wearing castings are not held in the machine by a single bolt, but simply by gravity alone, hence are easy of removal when worn too thin for further use. It can be set up and run in a few hours after arriving on the ground, and requires no expensive foundation. To run the machine, 12 to 15 H.P. is required. The weight of the machine, including frame complete, is less than 7000 lb., and the weight of the heaviest piece is 855 lb. The weights of the wearing parts are :—1 outer ring, 765 lb. ; 1 disc-plate 655 lb. ; 12 rolls or rings, 715 lb. ; total, 2135 lb.

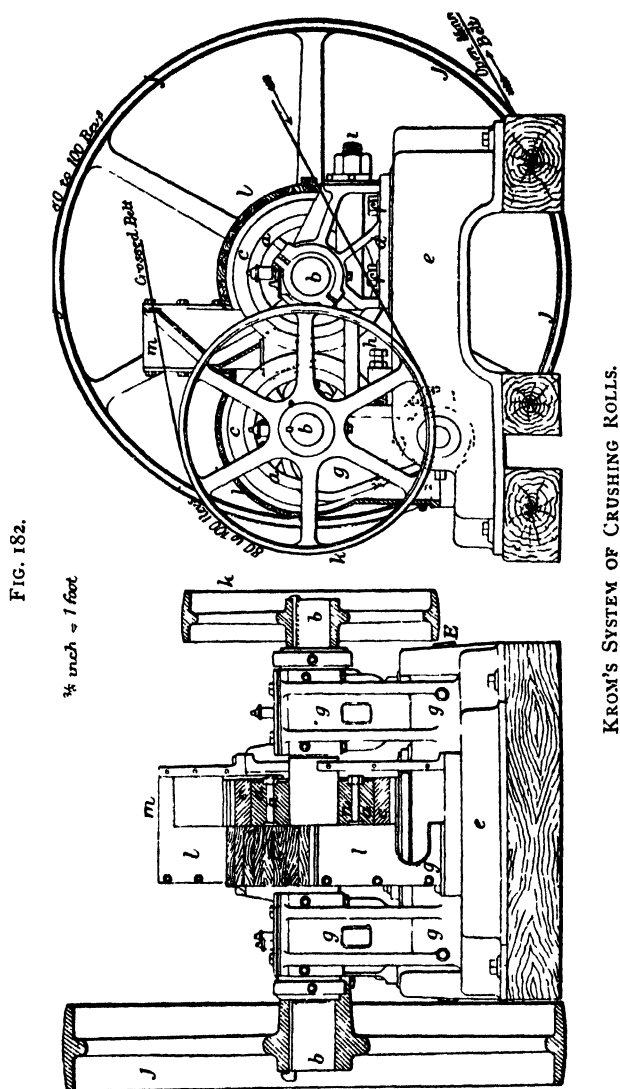
Lucop's.—It is claimed for Lucop's pulveriser that it will reduce 25 cwt. of quartz per hour with a 4 H.P. portable engine working at 40 lb. pressure on the boiler and 150 rev. of crank-shaft, while the first cost of the D size machine is but 150*l.*, and the repairs come to less than 3*l.* per ton dealt with.

Thompson's.—The action of Thompson's pulveriser consists in the use of a heavy ball within a revolving drum, the ball being thrown by centrifugal force against the material to be crushed. The size weighing 5 tons, and running with a 190 lb. ball, requires 10 H.P., and pulverises 60 tons per 24 hours to a degree that allows it to pass through a No. 60 screen.

Crushing Rolls.—Since the introduction, in 1882, of steel crushing rolls in the place of stamps, in the Bertrand Mill, Nevada, attention has been concentrated on this method of reducing ore to a fine state. This looked somewhat like a return to obsolete methods, for the modern stamp battery was currently supposed to be immensely superior to the old Cornish rolls. But when mechanical genius was directed to improving upon the Cornish rolls, it soon became evident that there was very much to be said in their favour. The recognised leading pattern of modern rolls is that introduced by Stephen R. Krom, of New York, and made in England by Bowes Scott & Read, Westminster.

The machine, as constructed by them, consists of two sets of cast-

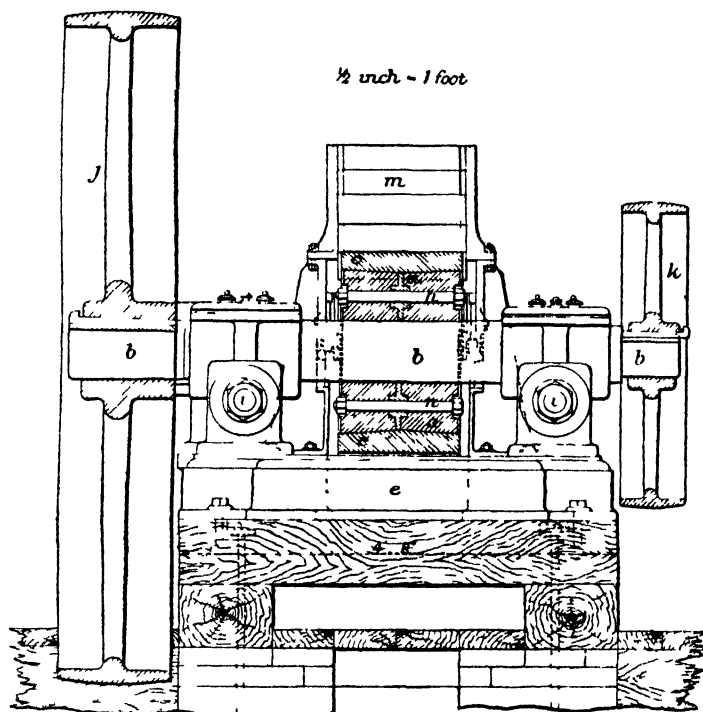
iron rolls *a* on steel axes *b* carrying steel tyres *c* (see Figs. 182, 183). The rolls vary from 26 in. to 30 in. in diameter, including the steel tyre. The tyres are made of the best open-hearth steel, and are $2\frac{1}{2}$ in. thick on the 26-in. rolls, and $2\frac{3}{4}$ in. thick on the 30 in. rolls, and can be worn down to $\frac{1}{2}$ in. with safety. They have been worn as low as $\frac{1}{4}$ in., but



there is danger that they will then spring and become loose. The pillar block *d* of one of these rolls is firmly bolted to bedplate *e* by nuts *f*. The second roll is set in a swinging pillar block, fixed in two strong

reinforced cranks *g*, which rotate in a journal *h* set in the cast-iron frame *e*. These double cranks are of exactly the same width as the roll. The shaft which supports them is 11 in. in diameter. Their journals are therefore pivoted with those of the rolls, and the two are thus forced to move together, so that this roll and shaft are always parallel to the other roll. The distance between the two rolls is regulated by screws *k*, one on each side, with jam nuts to prevent any motion after they are once set. The rolls are held to their position by two heavy bolts *i*, also

FIG. 183.



KROM'S SYSTEM OF CRUSHING ROLLS

with jam nuts, so that the position being once fixed, the distance of the surfaces cannot be changed by any action of the machine but the wear of the rolls. Formerly the distance was fixed by having the pillar boxes slide on the bedplate, but this was found not to work well, as they constantly became loose and thus pounded on the bed. It was also found almost impossible to get the surfaces of the two rolls parallel, and they wore unevenly. To the axis of the fixed roll a large driving wheel *j*, 7 ft. in diameter and 15 in. wide, is keyed, and to the movable one a smaller wheel *k*, 50 in. in diameter and 8 in. wide, is keyed. The

rolls are covered with housing *l*, to which an exhaust fan is attached, so that no dust escapes into the air; this protects the journals of the machine from abrasion. This housing carries a feed box *m*, arranged with a series of inclines so as to spread the ore in a continuous and even sheet between the surfaces of the rolls. Magnets are provided in the feed box to catch any pieces of steel which may be in the ore from broken tools, and which might indent the surface of the rolls.

The whole machine is self-contained and very compact, a pair of 26-in. rolls occupying a ground space of only 7 ft. by $7\frac{1}{2}$ ft. The tyres, which for the 26-in. roll weigh 816 lb. each, are held in place by two cast-iron heads, which are slightly conical in shape. One of these is shrunk on to the shaft and permanently fixed, the other is slit on one side and slips on to it. Both the heads are so placed on the shaft that the smaller diameter will be towards the centre. The steel tyre is turned out on the inside to correspond to this, so that it can be easily slipped over the permanent head and the loose one brought up to it. The two are securely fastened together by bolts *n*, so that when the movable head is drawn up to the permanent one, the slit in it closes, and makes it perfectly tight on the axle. It has been found expensive, on account of turning the inside of the tyre, to have it all made in one piece, so that it is sometimes cast in two pieces, which are turned to fit together in the centre. The bolts *n* draw them together perfectly tight. It was thought at first that the steel would wear unevenly on the two parts of the tyres thus joined, but it does not, and the economy and convenience realised by the construction is great. When the tyres are worn so thin as to be in danger of springing, they can be very easily removed and others substituted for them. To prevent delay, it is a good plan in every mill to have extra hubs with the tyres on, so that there may be no delay when the tyres are worn out. The worn-out tyres can then be replaced at leisure. In order to provide for abrasion on the sides of the machine, check pieces of composition metal are provided, which can be easily taken out and replaced when too much worn. The putting on of a new set of tyres requires but a short time, so that the delay when the rolls are to be replaced need be only very short, more especially if extra sets of rolls are kept on hand with the tyres already shrunk on the hubs, so that they may be adapted at once. It has been theoretically objected against the use of rolls, not only that they might wear unevenly, but that any stoppage would involve suspension of the whole work, while the stopping of the stamps would involve the suspension only of that particular head or battery, which could be repaired at leisure. This objection, however, in practice falls to the ground, for it has been found that repairs to the rolls can be very quickly made, and that stoppages are much less frequent.

Krom's rolls are generally arranged in double sets, one of which receives the coarse ore from the crusher and delivers it on to a screen of a determined size. That portion which does not pass through the screen is carried back to the first rolls. What passes the screen falls upon the second pair and from there to another screen. Part passes this last screen and goes on for treatment; part remains and goes back to the second rolls again. Whenever the screens show any wear in the rolls, the latter are stopped and reset. When very fine crushing is required, three pairs of rolls can be used, but generally two only are required. For quantities of 10 to 20 tons a day, a single roll can be used.

Krom's rolls are made of three sizes, with varying lengths and diameters. These are 22 in., 26 in., and 30 in. in diameter, and 14 in., 15 in., and 16 in. long. It has been suggested that they should be made longer, but there is no practical advantage in making them so, while the difficulty of fitting and making the tyres would be very much increased. In considering their efficiency and capacity, it must be borne in mind that the whole of the surface of the faces of the rolls is fitted to act on the ore, since it is even and parallel, and wears so. The ore escapes from the rolls by gravity as soon as it is crushed fine enough to fall through the space between them. If the feeding is automatic, or even when it is not, no clogging of the ore is possible. In order to form a comparison between ores and stamp rolls, the surfaces actually in contact at any given moment must be compared. A 30-stamp mill, with stamps weighing 750 lb., dropping 90 times a minute, with shoes and dies 8 in. in diameter, have, in round numbers, 50 sq. in. of surface on each stamp, or $50 \times 90 \times 30 = 135,000$ sq. in. of surface acting on the ore every minute. It may be assumed that two sets of 22-in. rolls will have an average diameter of 21 in. If these rolls make 80 revolutions per minute, and are 14 in. on the face, they will have a contact surface of 141,120 sq. in. per minute, or a little more than a 30-stamp mill. If the number of revolutions is increased to 100, the capacity is brought up to 171,000 sq. in. of surface, a little less than a 38-stamp mill, whose capacity is 176,400 sq. in. of surface. If the diameter of the rolls is increased to 26 in. with 15 in. of face, the average diameter may be considered as 24 in. If they make 80 revolutions a minute, the face surface will be 162,800 sq. in., equal to the surface of a 36-stamp mill. If the number of revolutions of this same roll are increased to 100, the surface capacity is the same as that of a 48-stamp mill. If the diameter is increased to 30 in., taking the average as 28 in. and the length of face 16 in., and the number of revolutions 80, the surface capacity will be equal to 47 stamps, and at 100 revolutions to 60 stamps. If, in each of these cases, where two sets of rolls have been considered, a third set is added, the capacity will be increased 50 per cent., and if four sets are

used, doubled. These calculations are made on the supposition that the surfaces acting are of equal efficiency in both rolls and stamps, and are, for that reason, more favourable to the stamps than actual practice shows. With rolls properly constructed the pressure is constant at each instant of time, while with stamps, on account of the varying height of the ore in the mortar, the cushioning of the stamp, and the fact that the stamp must not only crush, but also force the ore through the screens, it never can be constant. Actual practice in mills has shown that the output of rolls is always larger than that given by these calculations. That the work can be done as well with small rolls in a compact space, as with stamps which occupy a much larger one, is shown by the experience of mills where they are used; that they can crush as fine, is shown by the fact that rolls are now being used to crush after the stamps.

Power was formerly applied to rolls by gearing, which was constantly liable to get out of order, and limited the speed at which it was possible to run. Pulleys are now used, a belt passing over both large and small wheel. This allows of attaining a speed of 80 to 100 revolutions a minute, or higher if desirable, which was entirely impracticable with gear. The capacity of the rolls is thus increased, while the wear is confined almost entirely to the crushing faces. It does away with the noise of the gearing, and reduces the risk of breakage almost to nothing. Both rolls, when the machine is in operation, travel with the same speed of surface, but the small pulley is so speeded that when there is no ore between the rolls, it travels one or two revolutions per minute faster than the larger one. The reasons why this double system was adopted is that one of the rolls being movable and the other stationary, it would not be a good construction to place a large pulley on a movable pillar block. When ore is between the rolls, a single driven roll will cause the other to revolve, so that if most of the power to do the work is applied to the large wheel, only a small part of it will be necessary to ensure that the other roll will always bite, when fed with ore, and be kept in motion when the ore at any instant fails. It will be seen also that most of the strain of the rolls is taken upon the bolts *i*, and that, by the use of the two bolts *i* and *h* with the nuts *f*, all of which are provided with double nuts, no pounding action is possible. The defects of almost all the improved rolls has been that they were not provided for this pounding movement, and that they have been too light in weight to do the duty required of them. Both these defects have been remedied in this machine. The surprisingly small number of parts, all of which are easily accessible, their simplicity, as well as the small space occupied for such a very large output, and the possibility of adjusting the distance between the rolls at any time and in a very few minutes, makes this

machine, which is entirely self-contained, not only most efficient but very economical.

It has recently been shown by Professor Egleston that two sets of 26-in. rolls at the Bertrand Mill easily crush 150 tons of hard ore in 24 hours, so as to pass a 16 screen. The Mount Cory Mill has crushed 50 tons in the same time through a 30-mesh screen. The usual capacity for the best stamps on the same kind of ore is 2 tons per stamp, which make the rolls equal to a 50-stamp mill. At the Bertrand Mill, 9000 tons of ore were crushed without costing a dollar for repairs, with the expenditure of less than one-half the power required to do the same work in a stamp mill. At this mill 15,000 tons of ore passed the fine crushing rolls before new tyres were necessary. After crushing 20,000 tons, the coarse crushing tyres were still good for two months' wear. It is considered that each set of rolls with a set of repair linings of composition metal, will be capable of crushing 20,000 tons; the only expense for repairs being the tyres and the check pieces. There is no absolute limit to the fineness of the crushing.

The quantity of material which can be pulverised in any given time will depend on the amount of actual crushing surface which is in contact at each moment of time. It is evident that that machine will be the most efficient which presents the greatest amount of crushing surface at every instant of time. The cost depends on the amount of power required to do the work, and on the expense of repairing the wear and tear of the machine, and keeping it up to its greatest efficiency. The cost will be greatest when any part of the power is badly expended or wasted, and when the number of parts to the machine is so large as to make the construction or repairs expensive, and when for any reason skilled labour is difficult to obtain. For comparison, suppose the rolls have been used and slightly worn to a diameter which will represent the average wear of a 26-in. roll, and that the rolls are 15 in. long, running 100 revolutions per minute. The crushing surfaces of two such sets of rolls in actual contact at every instant of time will be equal to that of a 50-stamp mill with shoes and dies 8 in. in diameter, making 90 drops per minute. The action of the rolls is continuous. They have nothing to do and do nothing but crush the ore which falls between them. If during any instant of time there is no ore, the power expended by the machine is stored up in the flywheels. The power of the stamp is spent in first crushing the ore and then forcing it out of the screens. As the whole of it cannot be forced out at every blow, a considerable quantity of it will be acted on several times without passing the screen, and without being reduced any finer, and this ore not only consumes the force of the blow of the stamp, but cushions it, and thus still further reduces its effectiveness. The weight of the stamp and the height to

which it must be lifted are constant quantities, hence the power required to move it will be constant, while the effectiveness of the blow will depend on the size of the ore from the crusher, the quantity of it in the mortar, the size of the screen, and the effectiveness of each blow to force the crushed material out, which takes as much time as it does to crush it. No careful experiments have as yet been made to find the actual amount of power exerted in a stamp mill to crush the ore only, but the stamp has so much else to do that it is safe to say that it does not exceed 50 per cent. of the power actually expended. With the rolls held rigidly together the action is constant at any given time. The power, if not effective, is stored, and no piece of ore, not reduced, remains between the rolls; it falls by gravity, and is returned to the roll, if too large, by an elevator, so that no cushioning is possible. Stamps have a maximum of velocity which is very soon reached, beyond which the fall is reduced. It may be carried so far that the stamp head may be kept suspended in the air, and consequently do no duty whatever; while the increased speed of rolls always gives increased output, which beyond a certain limit is gained at the cost of the expenditure of power, but there is no cessation of action possible, as in the stamp. If we make a comparison of parts required in the construction of each machine, we shall find that the two machines of the same capacity will have the following working parts. For the rolls—

3 shafts.
4 journals.
4 pillar blocks for journals.
2 crushing rolls.
2 side plates as repair pieces.
12 wearing surfaces for each two sets of rolls.

27

All of these parts are large, made so strong that they are not liable to break, and the wear is not rapid. They can all be easily replaced, and all are easily accessible at any time. The stamp mill has

15 journals.
5 shafts, 10 stamps to a shaft.
5 cam shafts.
15 bearing boxes for the journals.
50 stems.
100 guide-boxes for stems.
50 stamp heads (bosses).
50 shoes.
50 dies.
50 cams with keys.
50 tappets.

440

All these parts are subjected to rapid wear, many of them are liable to break, most of them can be easily replaced, but the greater part of them are small and require much time to watch them, and if left to take care of themselves may cause much damage and stoppage in the machine. While most of them are accessible to make repairs, one or more of the stamps, or even batteries, must be hung up, and while any single repair does not consume much time, the multiplicity of them in the course of a year, does. The screens have not been considered, as they are common to both machines, although as they receive part of the blow of the stamp which thrusts the crushed ore against them with considerable force, they must wear more rapidly than in the rolls, where the ore falls on them only by gravity and for a very short distance. This simple comparison of the parts speaks for itself.

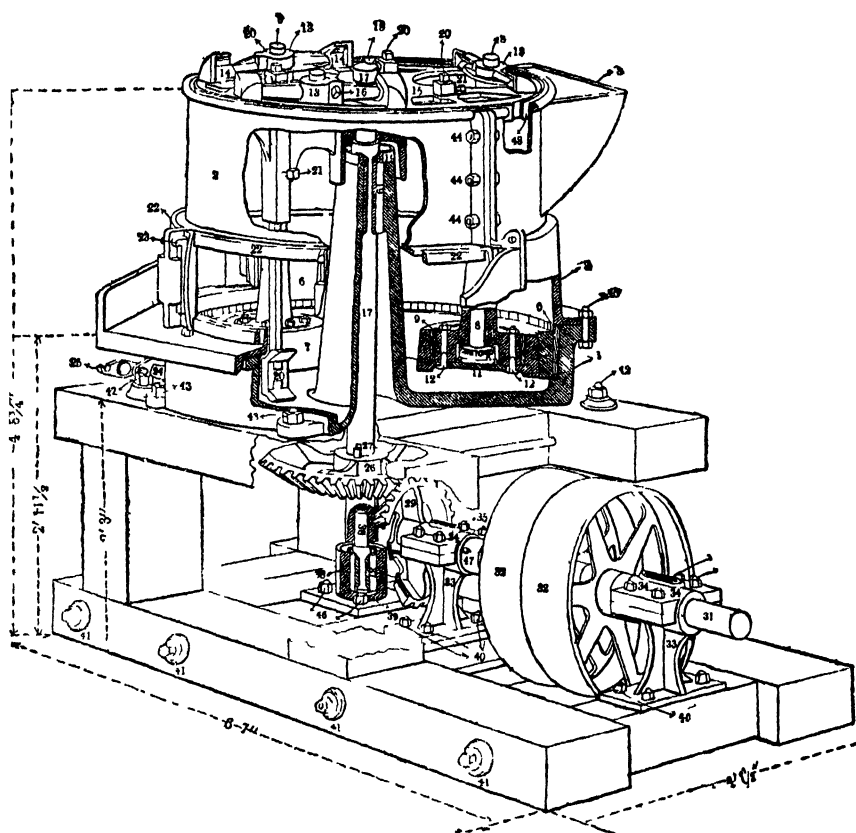
It has been found by experience that the condition of crushed ore from rolls makes the pulp much better suited for lixiviation than that from stamps, as the particles of ore coming from rolls are not only more uniform in size, but contain much less dust, which interferes with the rapid and effective passage of the solutions used. It has been found, too, that in chloridising, great fineness of the ore is entirely unnecessary, except in the rare case of the precious metal being very finely distributed through the gangue, so that with the exception of this single case, rolls seem to give the most favourable results. This is of importance in view of the favour which is being extended to chloridising processes.

The wear of rolls is almost exclusively confined to the steel tyres and cheek pieces ; that of stamps to the great number of parts already enumerated. Something over 80 per cent. of the steel of rolls can be safely used in crushing before it is necessary to put in a new roll. Generally it is not safe to run more than 50 to 60 per cent. of the shoes and dies before replacing them. On account of the number and complicated nature of the parts of the battery, skilled labour has to be used to a considerable extent, while only a nominal amount of it has to be used with the rolls. Every one who has ever seen a mill, knows how often the stamps have to be hung up, while rolls, if they are made strong enough at the outset, require little or no repair except the change of tyres, which is rapidly done and occurs at long intervals.

Centrifugal Roller Mill.—The centrifugal roller quartz mill, designed by F. A. Huntington, of San Francisco, and manufactured in England by Davey, Paxman, & Co., Colchester, is shown in Fig. 184. It is designed on an entirely different principle from its predecessor, the stamp battery. The illustration represents a 5-ft. mill. At the upper part of the figure will be noticed a circular horizontal frame of cast-iron, keyed at its centre to a vertical spindle G. This frame or "disc driver"

carries the rollers and scrapers, and has shrunk on its periphery a wrought-iron ring B. The method of suspending the rollers and scrapers will be best understood by the plan of the disc in Fig. 185. Round the edge of the ring four pairs of pockets P, are cast with it and project above it. Into each pair of pockets are set journals to form bearings for

FIG. 184.



HUNTINGTON'S CENTRIFUGAL ROLLER MILL.

the ends of the yokes, one of which is shown at Y. In Fig. 186, one of the rollers and yoke is shown in sectional elevation. It will be seen that the journals of the bearings at each end of the yoke are fixed into the pockets. This allows the rollers to swing in a radial direction with reference to the vertical spindle G, and, when in action, centrifugal force carries them hard against the steel ring die fitted on the inside of the rim. Each roller consists of a steel ring R (Fig. 185), 13 inch bore, and $1\frac{1}{2}$ inch thick. An internal flange on the ring forms a seat on which the

hub C rests. There is $\frac{1}{2}$ inch space all round between the hub and the steel ring, into which are driven the wood wedges W for securely fixing the ring to the hub. In addition to this, the ring is supported at the flange by two claw bolts passed through the hub. A hollow lubricating chamber H is left in the hub C, and a slot in the side of the roller shaft S forms a channel of communication from the chamber to the top, where oil may be added. This hole is kept plugged to prevent the entrance of grit. The lubricating chamber is hermetically sealed by a faced plate J, and rubber joint, rendering impossible any ingress of oil into the pan or amongst the amalgam. The roller shaft S is of steel and firmly keyed

FIG. 185.

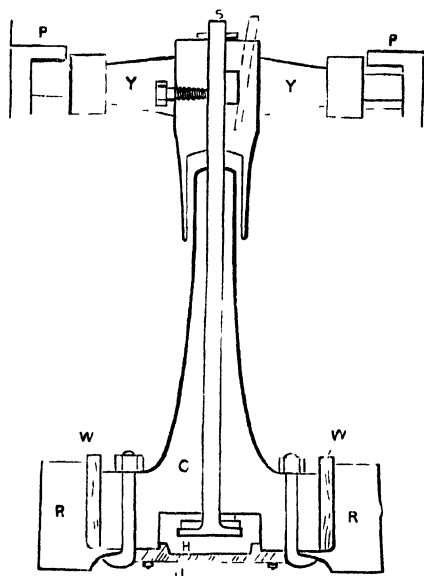
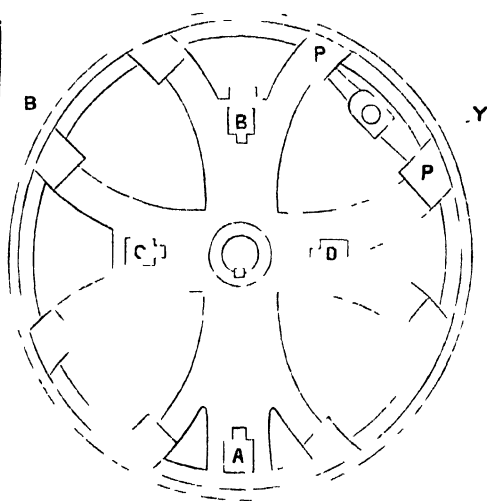


FIG. 186

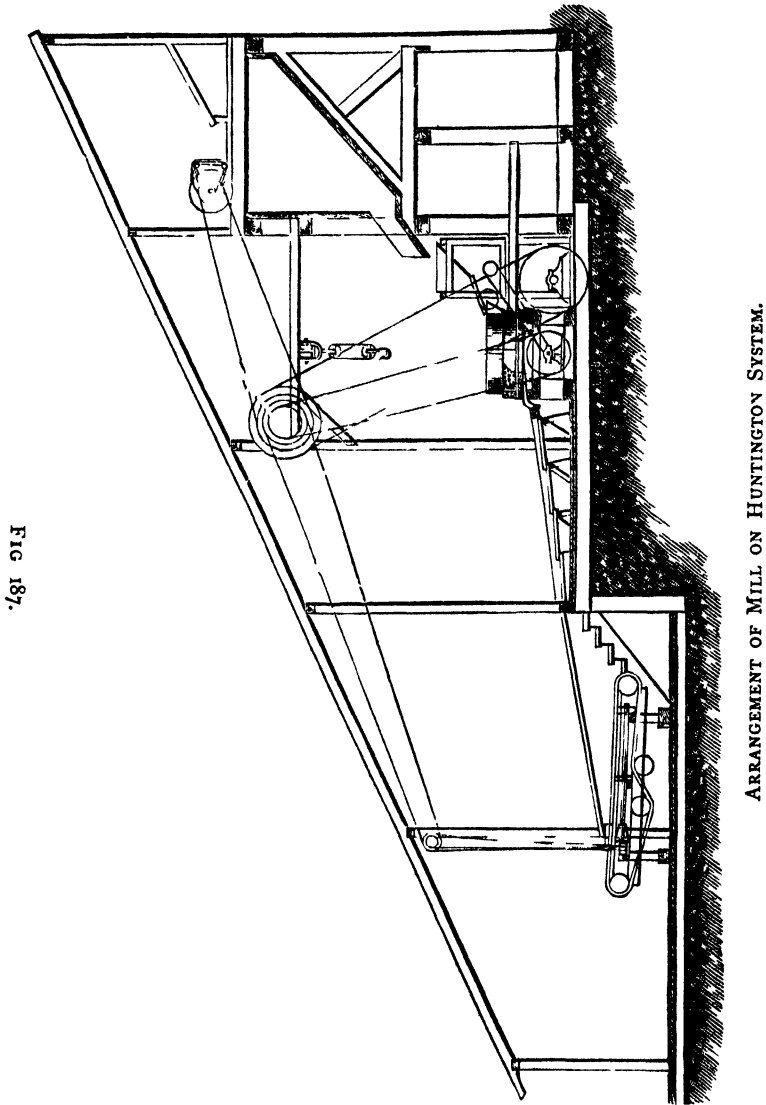


DETAILS OF HUNTINGTON'S MILL.

to the yoke at the top : it does not therefore rotate on its own axis, but is the centre round which the hub and rollers rotate. The weight of the roller is supported by the circular steel head at the lower end of the shaft. The rotation of the rollers on their own axes is caused by the centrifugal pressure against the ring die when the mill is in action. A clearance of one inch is left between the lower edge of the rollers and the bottom of the pan, so that the former pass freely over the amalgam without coming in contact with it, or in any way flouing it, whilst at the same time there is sufficient agitation to render the amalgamation perfect. Referring again to Fig. 186, the wooden supports for the scrapers are driven and keyed into the holes A B C D, which are at different

distances from the centre. The iron scraper fixed at the lower end of the wood carrier can be seen in Fig. 185 at F. By fitting these at different distances from the centre, the action of the scrapers tends to keep the quartz against the steel ring die. The outside shell enclosing the mill is a casting, the upper part being in two pieces, bolted together by vertical flanges. The whole of this shell is again bolted by a horizontal flange at its lower end to another casting of ring shape. This latter contains, wedged inside it, the steel ring die, against which the rollers bear. To render this part of the machine in every respect equal to the wear and strain, and prevent fracture of the casting when the ring is wedged in, a wrought-iron band or ring is shrunk on externally. The action of the mill will now be readily understood. Power being applied to the driving pulley, the countershaft is driven and the horizontal motion transmitted to a vertical motion by bevel wheels under the framework of the mill. The central vertical spindle is therefore rotated, carrying with it the disc driver, rollers, and scrapers. The quartz or material to be treated, being fed into the pan with water, at the hopper A, is immediately thrown against the steel ring, where the rollers come in contact with it and pulverise it. The precious metal in a free state at once amalgamates with the mercury run in the bottom of the pan (about 40 to 60 lb. of mercury are required in a 5-ft. mill), and the pulverised quartz, as soon as fine enough, is discharged through gratings extending along half the circumference of the mill. These gratings or perforated plates are made in various degrees of fineness to suit any required fineness of discharge. The delivery is both rapid and perfect, producing the fine grit without slime, which is then conducted to the distributing boxes, and passes over the concentrators in the usual manner. About 75 per cent. of the precious metal is retained in the amalgam. For the purpose of comparing the first cost, saving in transit, and labour of installation at the mine, let us take a ten stamp battery, whose output per hour is somewhat less than a 5-ft. Huntington mill. The ten stamp battery of 850 lb. heads, together with its boxes, frames, foundations, driving gear, &c., when erected, will cost 650% and weigh from 12 to 14 tons, in addition to which the cost of transit alone, in such districts as the Transvaal for instance, will cost nearly as much as the mill itself. Now a 5-ft. Huntington mill weighs only $5\frac{1}{4}$ tons, including frames, gear, &c., and the cost, including foundations and fixing, is but 350%. A most important matter is the time taken in preparing the foundations and fixing the mill at the mine. A Huntington mill can be completely fixed by two men in 15 working hours, whereas under the most favourable circumstances the ten stamp battery would require at least four men for 30 days, and frequently take many months before it is got into action. Even when started, it is often found that the foundations are

not satisfactory, causing further delay. Again, as regards the commercial efficiency of the two mills, the Huntington stands a long way ahead, for while working at a higher rate of output than the stamp battery (in



some cases from 10 to 50 per cent. higher), it absorbs considerably less than half the driving power. The cost for renewals of rings and rollers in the Huntington mill is about equal to that for dies and shoes in the stamp battery, but the latter has frequently against it heavy items of

expense for breakages and renewal of wearing parts in the stamps, which are entirely avoided in the centrifugal mill.

Fig. 187 illustrates a mill arranged on the Huntington system by Calvert, Cornes, & Harris, 30, Walbrook, London.

ARRESTING THE METAL.

General Principles.—The operations described on pp. 423–73 have had for their object the disintegration of the material containing the gold, in order that the latter might be liberated from the worthless material accompanying it in nature. The next step is to arrest and recover the valuable portion thus set free. This valuable portion is not all of one quality, but consists of two classes of material: one being gold in a clean, uncontaminated, “free” state; the other composed of various metallic sulphides (of iron, copper, lead, zinc, antimony, &c.), commonly known generically as “pyrites,” having more or less gold concealed within and between its crystals or grains. The separation or elimination of the valuable ingredients of the mass, the free gold and the pyrites, is effected by a combination of two distinct processes. The first consists in directing the pulverised or comminuted mass of ore through, over, or among a body of mercury, which metal possesses the peculiar property of absorbing all particles of free gold with which it comes into sufficiently close contact. This absorption is known as “amalgamation,” and as it constitutes the most certain and satisfactory method of collecting the minute particles of gold, from which the mercury can afterwards be separated without loss, and used an indefinite number of times, the object of all preliminary operations is to prepare the gold for amalgamation. The second half of the twin process of arresting the metal is the provision of a number of checks or obstructions to the onward flow of the matters leaving the reduction apparatus, with the object of presenting abundant opportunities for the valuable matters to avail themselves of their comparatively greater specific gravity to come to rest, while the worthless matters and a portion of the pyrites flow away in the current of water, to undergo further treatment afterwards.

The subject may be divided into the following heads:—(1) mercury applied in its ordinary fluid condition, (2) mercury applied as a coating on metallic surfaces, known as amalgamated plates, (3) special forms of amalgamator, for rubbing up the material with the mercury, and (4) blanket-tables.

Mercury.—Perhaps the commonest way of using mercury, and certainly by far the most general in the United States, is to supply it in regular or irregular quantities directly into the mortar or coffer of the stamps, so that it may be pounded up with the mineral. When this

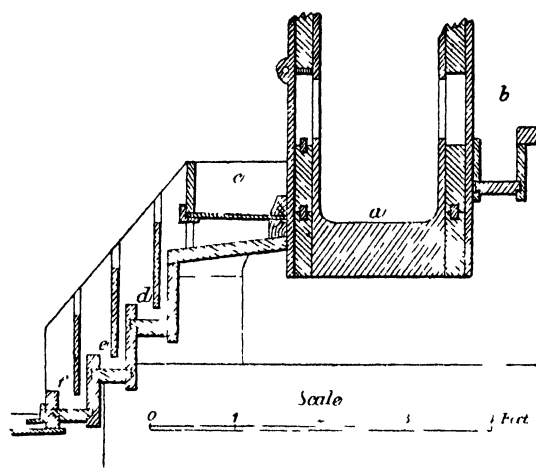
plan is adopted, amalgamated plates, to be described presently, are almost invariably employed in the mortar. The use of mercury in this manner would appear to be radically wrong. In the first place, the smashing and splashing of the mercury under the stamps must cause a considerable loss, for the simple reason that tiny particles of metal will be carried away by the force of the current, without ever having an opportunity to come into contact with the plates, or to deposit themselves subsequently. In addition to this, it may be broadly said that every gold-ore contains some pyrites. The stamping operation reduces this to a very fine state, and even liberates some of the sulphur to such a degree that it will form a coating over the globules of mercury and amalgam, which is technically known as "flouring" or "sickening," and by which the power of amalgamation with gold is destroyed, or reduced to a minimum. When amalgamation is alone depended upon for catching the gold, this circumstance will entail a triple loss,—a loss of gold which has escaped amalgamation, and a loss of both the gold and mercury which have become amalgamated, for the floured stuff, whether mercury alone, or mercury which has absorbed some gold, cannot be caught by contact with a fresh mercurial surface.

The second mode of exposing mercury to the crushed material may be divided into two heads, "ripples" or "riffles," and "troughs." Mercury-ripples consist of grooves cut across the ripple-board tables, inclined planes of wood, varying in length, placed in the route of the materials leaving the stamps. These grooves are cut about $2\frac{1}{2}$ ft. apart, and are 1 in. deep at the lower side, diminishing till they are flush with the surface of the bath at the upper edge, and about 3 in. wide. While at work, they are kept nearly full of mercury. They are generally used in combination with blanket-tables, and are most favoured in Australia.

The mercury-trough may also be considered as essentially Australian. A very effective arrangement of mercury-troughs and blanket-tables adopted by some of the largest Victorian companies, is as follows:—The material leaving the stamps is led into a trough, having a perforated plate at the bottom to keep back any coarse stuff, by which it is easily distributed; thence it passes into 3 connected troughs, containing mercury, dropping from the first into the second, and from the second into the third. Each of these troughs is fitted with a splash-board which, reaching down to within a certain distance of the bottom, compels the falling matter to penetrate the mercury more or less, before escaping over the lip of the trough. Each trough has a tap-hole on one side, by means of which the amalgam may be drawn off. The whole of the contrivance is under lock and key, to prevent stealing. At the end of the blanket-table another similar trough is placed, through which the

material passes before entering the waste-trough. The amalgam formed in all these troughs is periodically removed. Fig. 188 shows the exact arrangement adopted by the Port Philip Co., at Clunes:—*a*, stamper-

FIG. 188.

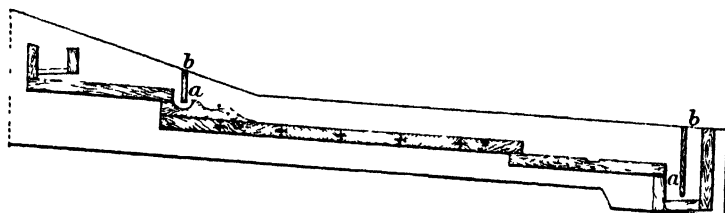


PORT PHILLIP MERCURY-TROUGHS.

box or mortar; *b*, back escape; *c*, perforated plate; *d, e, f*, mercury-troughs.

The Langlands Foundry Co. manufacture a ripple table as shown in Fig. 189, on a scale of $\frac{3}{8}$ in. = 1 ft. In the compartments *a*, mercury is

FIG. 189.

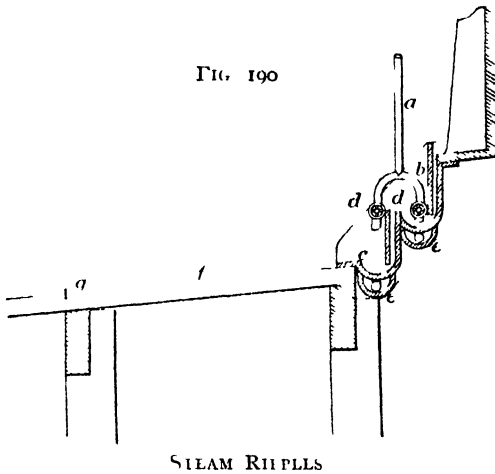


RIPPLE TABLE MADE BY LANGLANDS FOUNDRY

introduced, and the spurs *b* dip into the mercury troughs in such a manner as to compel all matters passing down the table to undergo actual immersion in the mercury baths provided at *a*.

It is well known amongst quartz crushers that mercury when heated has a greater affinity for gold, and acts much better as a gold catcher, than it does when cold. With a view to taking advantage of this property,

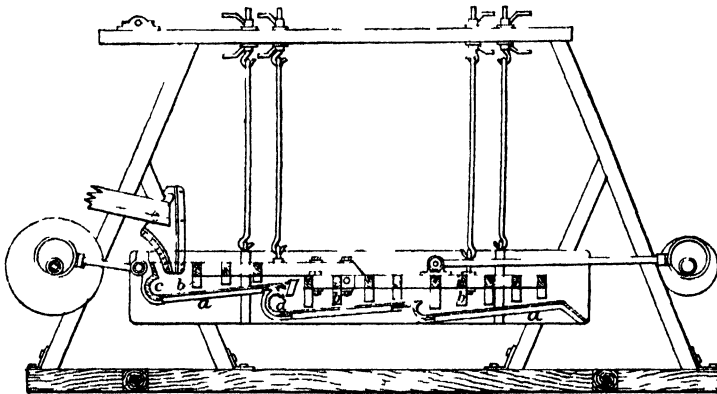
steam ripples are in use in New Zealand. These are made of cast iron, with a double bottom, as shown in Fig 190, the steam being introduced into the hollow part below where the mercury is placed. The cocks regulate the heat imparted, as the wider they are opened the greater will be the



discharge of the water from the condensed steam, and, consequently, more will be condensed and more heat imparted to the ripples. A steam supply pipe *a* conducts steam into the steam ripples *b c*, the supply being regulated by the cocks *d*, and the condensed water being drawn off from the jackets or double bottoms *e* by other cocks not shown; *f* are amalgamated copper plates, and *g* is a flat mercury trough, a series of which are, followed lower down by catch ripples and blanket tables.

Another contrivance for aiding the contact between the gold and mercury is the so-called "wave-amalgamator" introduced by Moon, and shown

FIG 191



MOON'S AMALGAMATOR.

in Fig. 191. This machine has a simple and easy swinging motion of 3 in. backwards and forwards, which creates an intermittent wave of quick-silver and water, into which the pulp from wet crushing battery (or other wet pulverising machines) is fed. A new surface of mercury is thus

constantly being presented to the pulp. The incline of the bottom *a*, instead of being towards the discharge end, is the reverse, in order that the free action of the pulp, mercury, and water, may be of sufficient duration to brighten, by attrition, the small particles of "rusty" gold; while the tendency of the light particles to be drawn into or around the wave of mercury caused by the drawback of the wave, should enable the finest gold to be saved. Inside the machine, running parallel with, and as near as may be convenient to the inclined bottom *a*, is placed a series of scrapers or rakes *b*; by this means the heaviest ores are prevented from "packing," and assistance is rendered to the water in carrying off the tailings.

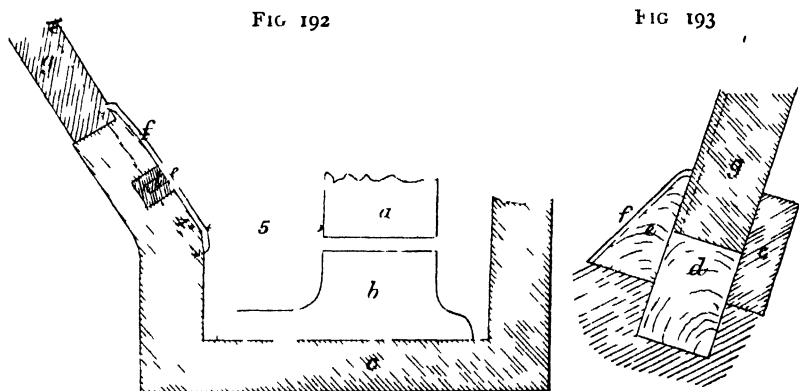
To work the machine, care must be taken to level the bed plates, which are made of wood. About 60 lb. of mercury is placed in the first circular back *c* nearest the feed end, and a similar quantity can be placed in the second circular back *d*, if it is desired to amalgamate the tailings from the first. So long as the mercury is not too heavily charged with gold, and provided the pulp is regularly supplied, the process should go on without attention. The speed of the amalgamator should be about 76 strokes per minute. But little power—from $\frac{1}{2}$ to 1 h.p.—is required. As regards capacity, one machine, smallest size, under ordinary conditions, is sufficient for a 5 stamp battery, equal to 10 tons per 24 hours. This size is recommended on account of convenience of transportation; they can be made of three sizes however, to treat 10, 15 or 20 tons per 24 hours. The amalgamators can be readily placed in position, and be ready to work within a few hours after their arrival at the mine; they will run in conjunction with any existing machinery. An ordinary mechanic can put them up. The size of a 10 ton machine is about $2\frac{1}{2}$ ft. by 9 ft. The machine can be boxed in, in a space 12 ft. by 5 ft.; or two, in a space of 12 ft. by 8 ft., so as to allow no access save under lock and key.

Amalgamated plates.—Amalgamated plates are prepared by a somewhat delicate and tedious process of covering one side of pieces of sheet copper with a coating of mercury. That portion of the interior of the batteries which is not occupied by the screens is lined with these plates, fixed in an inclined position, and so as to be readily removed and replaced. By the churning that takes place in the battery, particles of gold, mercury, and amalgam are splashed upon these plates, and attach themselves to the surfaces, which are periodically cleaned. Outside the batteries are placed tables, covered with similar amalgamated plates, adjusted at such an inclination as will permit a ready flow of the materials over the surface, without being so rapid as to wash away the gold and amalgam, or prevent their adhesion to the plate. The inclination necessarily differs according to the supply of water, and other conditions.

The gold and amalgam collected on these plates are removed in the same way as from the others.

There are two ways of fixing these copper plates in such a position as will bring them into constant contact with the crushed ore and the thereby liberated particles of gold. Firstly, the mortars are cast purposely to receive, on proper shelves, this copper-plate lining; and, secondly, the old boxes may be at once adapted for the same purpose until a renewal is necessary, when the first-named should be obtained and no other. It may be mentioned in passing that such mortars prevent any possibility of peculation of amalgam.

The first-named kind of mortars (Fig. 192) are cast to a pattern, so as to introduce beneath the rim for the gratings (inside the mortars) a kind of sloping shelf 4 in. wide for the whole length of the front discharge at an angle of 30° to 35° towards the false bottoms. Into this shelf, 4 holes



FIXING AMALGAMATED PLATES.

e are drilled, or recesses cast $\frac{1}{2}$ in. in diameter and $1\frac{1}{2}$ in. deep, which are plugged with dry and soft wood. A copper plate of the exact size of this shelf, $\frac{1}{4}$ in. thick, is laid on a strip of blanket equal in size, and 4 copper screws are inserted through corresponding holes $\frac{1}{8}$ in. diameter, in the copper-plate and blanketing into the wooden plugs, which, on getting wet, will swell, and thus the plates are securely screwed down until the next cleaning-off, when they are unscrewed, and so forth. Both the upper and lower edges of the plates should receive a batter, in order to make a good joint, and to prevent the finely-crushed ore getting behind. Any subsequent repairs should be made with copper rivets, and, in fact, the more battered these plates become, the better are they for the interception of gold. The explanations of the illustration are—*a*, shoe; *b*, die; *c*, box; *e*, holes plugged with soft wood; *f*, copper plate; *g*, grating-frame; *h*, line for grating; *z*, screw countersunk in copper plate.

The other method does not necessitate the immediate change in the pattern of the boxes in use until they are unfit for use, and the adaptation of old boxes for these copper plates does not interfere with their efficacy.

The ordinary frame *g*, which holds the gratings, is not quite so high, in order that a piece of soft wood *d* (Fig. 193) may be screwed on to its lower part; it is $3\frac{1}{4}$ in. high and $1\frac{3}{4}$ in. wide; and at the same time the cast-iron lug *c* is made higher by broad strips of boiler-iron, so as to catch both the wooden insertion and the grating-frame properly. At the inner side of the piece of wooden insertion *d*, a triangular strip of soft wood *e* is screwed on, which has the prepared copper plate *f* fixed to it, and in this manner the plates will act nearly as well as in the other case. The plates used inside the mortar are $\frac{1}{8}$ to $\frac{1}{4}$ in. thick, while outside $\frac{1}{16}$ in. sheet copper suffices.

When ordinary copper plates are to be employed, they are prepared in the following way:—The copper must be of the very best quality, and should not have been rolled to such an extent as to make the surface bright and hard, as it will not then possess the necessary porosity for absorbing the mercury. In such a case, it must be annealed before use, which is generally done by heating the plates beneath till wood will ignite on the upper surface, taking care that the heating is uniform, and avoiding oxidation of the upper surface during the process. The plate is then flattened by beating it with a wooden mallet with a flat slab to bear the blows, as direct blows would batter the surface. The fixing of the plates in the mortar has already been described; it may be noted, however, that the screws should be either of copper or extremely hard wood. When the plate is secured, its surface is dressed smooth with the mallet and slab, and then scoured with sand and wood-ashes, and rubbed perfectly bright with very fine emery-cloth; or it may be washed with a strong soda solution, which removes grease and imparts a bright face. Next it is washed with clean water, and rubbed over with a solution of potassium cyanide, in the proportion of $\frac{1}{2}$ oz. in 1 pint of water, taking care to finally wash with warm water, as any excess of cyanide would dull the surface. The bright clean plate is then ready for the mercury, and is rubbed with a mixture of fine mill tailings or other fine sand and powdered sal ammoniac, applied with a brush, while a little mercury is sprinkled on the surface so long as the copper absorbs it. The addition of the sal ammoniac to the sand for rubbing in removes the effects of oxidation, which sets in very rapidly on the exposed surface, and prevents amalgamation. The mixture is left on the plate for $\frac{1}{2}$ hour, when the latter is rewashed with cyanide solution, and more mercury is added until absorption ceases. Gold-amalgam (or, failing it, silver-amalgam) is then rubbed in with a piece of indiarubber belting, using sal ammoniac solution to keep the surface bright.

When electro-silvered plates are used instead of simple copper ones, they are prepared in the following manner. They are well cleaned and burnished with very coarse sand-paper, after being straightened out; a coat of good beeswax is then applied to the side not cleaned, in order to confine the process to one surface only. The plates thus prepared are hung in a bath containing a solution of silver of regulated strength, then connected with a battery, and thus electro-plated with pure or coin silver on one side, in such a manner and with such a quantity of silver as will not amount to less than 1 to 3 oz. per sq. ft.; any larger percentage is preferable, if the coating presents as rough a surface as possible. These electro-silvered copper plates are highly esteemed for their amalgamating powers, as the copper and silver form, as soon as the mercury is applied in the usual way (just described), a powerful galvanic battery, the action of which is much heightened by the slightly acid character imparted to the water by the crushing of pyritous matters. A well-known maker of these silvered plates is E. G. Denniston, 653, Mission Street, San Francisco. Many mills now silver their own plates. Silvered plates are fast replacing ordinary copper ones, being much more easy to keep clean.

The cost of the ingredients used for a 5-stamp battery in 10 months, at Californian prices, is thus given by Thureau:—

				\$	c.
6 Bunsen's elements	..	.	.	24	00
4 glass cylinders	..	..	..	3	20
5 lb. cyanide	..	..	.	4	25
1 porous cup	..	..	..	0	50
7 lb. nitric acid	..	.	.	1	15
50 lb. cyanide (fused)	.		..	37	50
14 lb. nitric acid	..	.	..	2	50
(14 <i>l.</i> 12 <i>s.</i> 1 <i>¼d.</i>)				73	10

The highly favourable opinions first expressed with regard to amalgamated plates have had to be modified by subsequent experience. The plates require constant attention, and to be kept free from film, which is liable to form on the surface, and will altogether prevent amalgamation if not removed. For this purpose, sal ammoniac, potassium cyanide, and ammonium chloride are used to wash the plates, they destroying greasy adhesions and dissolving the oxides of copper and other metallic salts caused by the action of the water employed in the battery, which almost always contains mineral salts in solution. All things considered, under ordinary circumstances, amalgamated plates probably never catch more than 55 per cent. of the gold-assay of the ore, and when other appliances are not adopted in conjunction with them, the loss of gold may amount to half the original quantity present. It may be safely said that their

use has been abandoned on all properties where any pretence is made of saving above 80 per cent. of the ascertained gold contents of the ore.

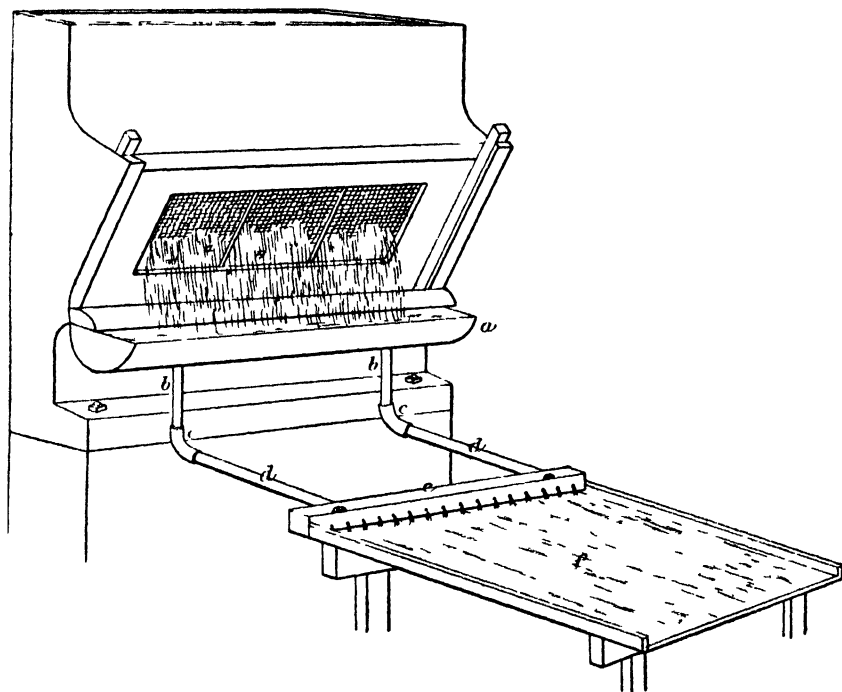
Raymond expresses himself in the following terms concerning the results of using amalgamated plates. The product of gold varies between 30 and 50 per cent. of that in the ore, and averages 40; 15 per cent. of the total gold remains in the unredced portion of the ore, and thus 45 per cent. actually freed still escapes the plates,—7 per cent. being dissolved in the $\frac{2}{3}$ of the mercury charged, which also escapes, and 38 per cent. escaping as unamalgamated gold. Of the amalgam obtained, the interior plates yield about 67 per cent., the outer 20 per cent., the skinmings 13 per cent.; or the inner plates yield $\frac{3}{4}$ and the outer plates $\frac{1}{4}$ of the amalgam obtained, or 30 and 10 per cent. respectively of the gold contained in the ore. Reckoning by units of surface, the inner plates collect 36 times as much gold as the outer. The amalgam is collected from the plates by “sweating” them, or preferably by immersing them in hot water till the amalgam is soft enough to scrape off.

From inquiries made of a number of Australian managers, the following inclines per foot are recommended for copper plates, viz. $\frac{1}{2}$ in., $\frac{3}{4}$ in., $\frac{7}{8}$ in., 1 in., $1\frac{1}{8}$ in., $1\frac{3}{8}$ in., $1\frac{1}{4}$ in., $1\frac{3}{8}$ in., $1\frac{7}{8}$ in., or an average of exactly 1 in. per ft. Bland finds amalgamated plates less efficient than mercury drops and blanket tables, and mentions that the incline depends greatly upon the supply of water—the smaller the supply the greater must be the incline. Clark states that copper plates are generally laid on the ripple tables between the wells of mercury, the object being to bring every particle of the crushed sand, as it escapes from the battery, in contact, as far as possible, with the surface of the mercury, or with the silvered surface of the copper plates. The incline of $1\frac{7}{8}$ in. has been found to work very well for all kinds of stone operated upon, just enough water being used to clear the table. Too much water is objectionable, as it is likely to carry away fine gold. Parker remarks that a great deal depends upon the width of the tables; if they are made the full width of the boxes, more water must be used, or the incline must be greater. Where tables are narrowed to suit the discharge of the gratings, the plate laid immediately under the lip of the box should not have a greater incline than $\frac{3}{4}$ in. per ft. The second plate leading to the blanket strakes should have an incline of 1 in. per ft., and the blanket table $1\frac{1}{2}$ in. per ft. If the tables are the full width of the boxes, the top plate should have $\frac{7}{8}$ in. incline, and the second plate $1\frac{1}{8}$ in. per ft. It must be understood that this last paragraph refers only to plates laid outside the mortar or coffer, as few if any Australian mills have plates inside the mortar.

At the Blue Bell mine, California, Superintendent Tregidgo has introduced an improvement, devised to increase free access to the front of the battery, as shown in Fig. 194. The device consists of a semi-

cylindrical iron trough *a*, the length equal to the width of the mortar, and having a breadth of 1 ft., and a depth of 6 in. This trough is fixed under the lip of the mortar, in the place usually occupied by the electro-plated apron; in the bottom of the trough are two openings 2 in. diameter, in which are fixed 2 pieces of 2-inch iron pipe *b*, each about 18 in. long, extending vertically downward. The lower ends of these

FIG. 194



BLUL BELL BATTERY FRONT.

pipes are connected by elbows *c* with horizontal pipes *d* of equal diameter, and about 3 ft. long. The trough receives the pulp from the mortar, the pipes into which the pulp flows deliver it to a distributing box *e*, from which it passes, through holes bored in the front, on to the apron *f*. By this contrivance the apron is removed some distance from the battery, in front of which the workman can pass without having to climb over the apron or sluice. When it becomes necessary to clean the mortar, the upper ends of the pipes are plugged, the trough forming a convenient receptacle for the rock, sand, &c., taken from the mortar. The sides of this trough are lined with curved plates of silvered and amalgamated copper, not fixed close to the side of the trough, but with a space of about $\frac{3}{4}$ in. left between. The pulp from the mortar swashes

from side to side in the trough in front of the copper plates, and a considerable portion of the gold is retained on the plates, which are secured to the trough by bolts with rubber washers placed between them, thereby preventing their contact. Tregidgo also uses a lip-plate on the mortar, outside of the screen. The inside back plates are 5 in. long by 12 in. wide; the front plates are attached to chuck-blocks as usual, the foot-rail of the screen frame being about 5 in. high, in order to afford the necessary space. The faces of the "chuck-blocks," and consequently of the plates, are rounded instead of being flat, the rounded form being better adapted for saving the gold. The aprons here in use are 5 ft. by 4 ft., with an inclination of $1\frac{3}{4}$ in. to the foot; at their lower end is a drop of about 1 ft. to the sluices; the pulp, however, does not drop, but flows downward on a curved plate which catches a good share of the amalgam.

Amalgamating Pans.—The term "pan" is used generically to denote a very large class of machines working on the same principle as the snuff-muller. Some are intended only to rub the ore against a surface of mercury, while others at the same time more thoroughly grind it. To the former class belongs the Hungarian bowl or Tyrolese mill, while the latter includes the *arrastra*, the Italian *molinari*, the Chilian mill, &c.

Pans present a great variety in the details of their construction. The common features are a round tub, usually of cast iron, but sometimes with wooden sides, 4 to 6 ft. in diameter and about 2 ft. deep, having a hollow pillar cast in the centre, within which is an upright shaft projecting above the top of the pillar that may be set in revolution by gearing below the pan. To the top of this shaft is attached, by means of a key or feather, a yoke or driver by which the muller or upper grinding surface is set in motion. To the bottom of the pan, on the inside, is fixed a false bottom of iron, cast either in sections, commonly called dies, or in one piece, having a diameter a little less than that of the pan, and with a hole in the centre adapted to the central pillar. This serves as the lower grinding surface. The muller, forming the upper grinding surface, is usually a circular plate of iron corresponding in size and form to the false bottom just described, having a diameter nearly equal to that of the pan, and a flat, conical, or conoidal form, according to the shape of the pan-bottom. Its under side is furnished with shoes or facings of iron, about 1 in. thick, that may be removed when worn down, and replaced by new. The muller is attached to the driver, which is put on and over the central pillar of the pan, and, being connected with the interior upright shaft as described, is thus caused to revolve. There are various appliances for raising or lowering the muller, so that it may rest with its whole weight upon the pan-bottom, in order to produce the greatest grinding effect, or be maintained at any desired distance above

it, when less friction or more agitation is required. Various devices are also in use for giving proper motion to the pulp, so that, when the muller is in revolution, the material may be kept constantly in circulation, passing between the grinding surfaces and coming into contact with the mercury. Some pans are cast with a hollow chamber, 1 to 2 in. deep in the bottom, for the admission of steam, in order to heat the pulp, while others employ only "live steam," which is delivered directly into the pulp by a pipe for that purpose.

The quantity of ore with which a pan is charged for a single operation, varies from 600 or 800 to 4000 or 5000 lb., according to the size of the pan. The ordinary charge of pans most generally in use at present is 1200 to 1500 lb.

In charging the pan, the muller is raised a little from the bottom, so as to revolve freely at first; water is supplied by a nose-pipe, and at the same time the sand is thrown into the pan with a shovel. Steam is admitted either to the steam-chamber in the bottom of the pan or directly into the pulp. In the former case, the temperature can hardly be raised as high as in the latter; but, on the other hand, when steam is introduced directly, care is necessary to avoid reducing too much the consistency of the pulp by the water of condensation. The pulp should be sufficiently liquid to be kept in free circulation, but thick enough to carry in suspension, throughout its entire mass, the finely divided globules of mercury. In some mills, both the methods of heating are employed in the same pans, the temperature being first raised with each charge by live steam, and afterwards sustained by admitting steam to the chamber only. Some pans are covered with wooden covers to assist in retaining the heat. When properly managed, the temperature may be kept at or near 200° F. (93° C.). When, in the use of live steam, the pulp becomes too thin, the supply of steam is cut off, the covers are removed, and the pulp is allowed to thicken by the evaporation of the water. The steam in the chamber may keep the temperature up to the desired point in the meantime. Another advantage of the steam-chamber is that exhaust-steam from the engine may be used in it, while for use in the pulp it is better and customary to take steam directly from the boilers, because that which comes from the cylinder of the engine is charged with oil and is injurious to amalgamation.

The muller is gradually lowered after the commencement of the grinding operation, and is allowed to make about 60 or 70 revolutions per minute. In the course of an hour or two, the sand should be reduced to a fine pulpy condition. When this has been accomplished, and by some mill-men at a still earlier stage, even at the beginning of the operation, a supply of mercury is introduced into the pan, the muller is slightly raised from the bottom to avoid too great friction, which would

act to the disadvantage of the mercury, and the action is continued for 2 hours longer, during which the amalgamation is in progress. The mercury is supplied by pressing it through canvas, so as to scatter it upon the pulp in a finely-divided condition. The quantity varies greatly in different mills, the ordinary supply being about 60 or 70 lb. to a charge of ore consisting of 1200 or 1500 lb. In some mills, a quantity, varying from 75 to 200 or even 300 lb., is put into a pan when starting after a clean-up, and subsequently a regular addition of 50 or 60 lb. is made with each charge.

Two hours having been devoted to the grinding, and two or three more to amalgamation, the pan is discharged, and its contents are received by a settler or separator. The discharge of the pan is usually aided by a supply of water, which dilutes the pulp and permits it to run freely from the pan into the settler. The pan, being emptied and partly washed out by the stream of water, is again charged with a fresh quantity of sand, and the grinding operation is resumed without delay.

The main objects sought for by inventors of pans have been to produce grinding surfaces of most effective form, securing the greatest uniformity of wear with economy of power; to obtain the most favourable conditions for amalgamation, depending mainly on the free circulation of the pulp, the uniform and thorough distribution of the mercury, and the proper degree of heat; and to combine with these requirements, simplicity and cheapness in construction, facility in management and repair, large capacity, and economy of time, labour, and materials in the performance of duty.

The attempts that have been made to obtain these results have met with varied success, the different devices of any one pan sometimes obtaining a high degree of excellence in certain details at the cost of it in others.

Among the differences in characteristic features of pans, the most noticeable is that of the bottom and the grinding surfaces, some being flat, and others variously curved; other details, of more or less importance, such as the construction of the muller and the method of attaching it to the driver, the form of the shoes and dies, the means of fixing them in place, of providing for the heating of the pulp, and for its circulation during the grinding and amalgamating process, vary considerably in the several patterns.

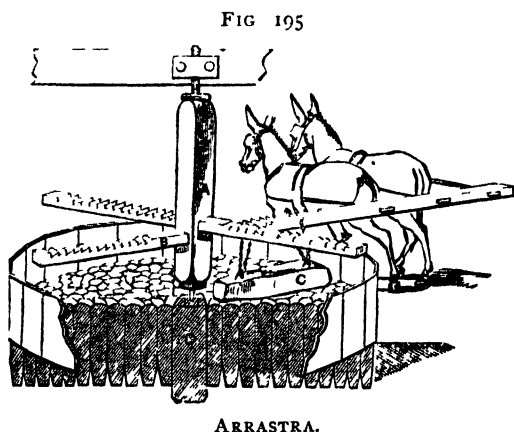
The opinions of practical mill-men are somewhat divided regarding the comparative advantages of the different forms of pan-bottoms. The prevailing opinion, however, seems to be, all things considered, in favour of the flat bottom. While other forms of grinding surface may possess superior advantages theoretically, their greater efficiency in practice is often lost by the unequal wear of the surface of the muller,

usually resulting from the difficulty of keeping the other parts of the machine, on which the grinding surfaces depend, in perfect order. The various parts of the flat muller are simpler in form, more easily handled, and more conveniently replaced when worn out. The flat-bottomed muller involves the expenditure of more power in carrying its load of thick pulp; but this disadvantage is counterbalanced, in the opinion of some, by the more complete distribution of the mercury, and the consequently more perfect amalgamation.

The various forms may now be considered in alphabetic order.

Arrastra.—The following description and illustration of a Mexican arrastra are taken from Phillips's well-known work.

"The arrastra consists of a circular pavement of stone, about 12 ft. in diameter, on which the quartz is ground by means of 2 or more large stones or mullers, dragged continually over its surface, either by horses or mules, but more frequently by the latter. The periphery of the circular pavement is surrounded by a rough kerbing of wood or flat stones, forming a kind of tub about 2 ft. in depth, and in its centre is a short wooden post, firmly bedded in the ground, and standing nearly level with the exterior kerbing. Working on an iron pivot in this central post is a strong upright wooden shaft, secured at its upper extremity to a horizontal beam by another journal, which is often merely a prolongation of the shaft itself. This upright shaft is crossed at right angles by two strong pieces of wood, forming four arms, of which one is made sufficiently long to admit of attaching 2 mules for working the machine. The grinding is performed by 4 large blocks of hard stone, usually porphyry or granite, attached to the arms, either by chains or thongs of raw hide, in such a way that their edges, in the direction of their motion, are raised about 1 in. from the stone pavement, whilst the other side trails upon it. These stones weigh from 300 to 400 lb., and in some arrastras 2 only are employed; in which case a single mule is sufficient to work the machine. Fig. 195 is a sectional view of a



Mexican arrastra, as usually constructed; in which A is the upright shaft; B, arms to which mullers C are attached; and D, the central block of wood in which the lower bearing works.

"Some of the arrastras used by Mexican gold miners, and for the purpose of testing the value of quartz veins, are very rudely put together the bottom being made of unhewn flat stones laid down in clay; but in a well-constructed arrastra, intended to be permanently employed, the stones are carefully dressed and closely jointed, and, after being placed in their respective positions, are grouted in with hydraulic cement.

"The charge for an ordinary arrastra is 450 lb. of quartz, previously broken into pieces of about the size of pigeons' eggs. The machine is now set in motion, a little water being from time to time added, and at the expiration of from 4 to 5 hours, the quartz has become reduced to a finely-divided state, and more water is added, until the contents of the arrastra assume the consistency of tolerably thick cream. Mercury is then sprinkled over its surface to the amount of $1\frac{1}{2}$ oz. for every ounce of gold supposed to be contained in the finely-divided rock, which is generally known, with a considerable degree of accuracy, from the results obtained from previous charges. The grinding is after this continued for another 2 hours, during which time the mercury is divided into minute globules, and becomes disseminated throughout the mass, which should be of such a consistency as not to allow it to sink to the bottom, but be so held in suspension as to meet and amalgamate with all the particles of gold. At the expiration of this time, the amalgamation is considered complete, and the process of settling the amalgam from the ground silicious matter is commenced. Water is now let into the paste, so as to render it very thin and perfectly mobile, the mules being driven very slowly, in order to allow the particles of gold and amalgam to yield to the influence of their densities, and to sink to the bottom. After having in this way slowly agitated the mixture for about $\frac{1}{2}$ hour, the thin mud is allowed to run off, leaving behind it, on the bottom of the arrastra, the gold combined with mercury in the form of amalgam. Another charge of broken quartz is now put in, and the operation is repeated, time after time, until it is thought desirable to stop for the purpose of cleaning up. The length of a run, or the period which is allowed to elapse between one cleaning up and another, varies according to circumstances. In the roughly constructed arrastra, having a bottom of uncut stones laid in clay, the run is seldom less than 10 days, and is sometimes extended to 3 weeks or a month. In this case the amalgam settles in the crevices between the paving stones, which have to be dug up, and all the sand and mud between them carefully washed. If, however, the machine be well constructed, and provided with a closely-paved bottom, the cleaning up is more frequently repeated, since the mercury and amalgam do not find their way so readily between the stones, but remain on the surface, from which they

are easily collected in an iron vessel, for subsequent treatment by straining and retorting.

"The arrastra does its work slowly, and consumes a large amount of power in proportion to the quantity of rock crushed, but is an excellent amalgamator, and is often valuable for the purpose of testing newly-discovered veins, and ascertaining their approximate yield. It is also the arrangement most commonly adopted by a miner, who, having found a rich pocket in his vein, is desirous of converting a portion of it into money, and of ascertaining whether it be likely to continue productive, before incurring the expense of erecting more costly and complicated apparatus. A modification of the arrastra is not unfrequently employed for the treatment of pyrites separated from tailings by washing, and is generally considered to be well adapted for that purpose."

Prof. Egleston remarks that, notwithstanding the extreme slowness with which the arrastra works, as against the rapidity of the stamp-mill and the pan, it has not been entirely replaced by them, and can often be used where, from local circumstances, the nature of the ore, or the limited amount of capital at the disposal of the miner, neither the stamp nor the pan and settler could be used. The idea of modern pan amalgamation was undoubtedly taken from the arrastra, which the early settlers of California found in use among the Mexicans. It combines most of the best points of the stamp, the pan, and the settler. But to the early miners, the advantages of its simplicity of construction, cheapness, and high yield, were as nothing compared with the necessity of making a large output in a short time, whatever the loss; and they thus modified it little by little, trying to keep its essential features and principles, until it disappeared entirely. It has real advantages, however, which prevented its being absolutely relegated to the catalogue of historical metallurgical appliances. Its cheapness and simplicity of construction place it so easily within the means of those who have but little capital to invest, and who, therefore, cannot erect a stamp-mill while prospecting, or even developing their claims. The economy with which it can be erected and worked, and the high yield which it gives of the precious metals when it is properly managed, more than counterbalance the deficiency of output, especially when the claims are not rich, or where they are not certain to last for any great length of time. It is especially adapted to the class of free-milling gold ores, which are generally found on the outcrops of veins or near the surface. Other classes of ores which are not free milling are not suited to it, nor to any other process which does not involve a much more expensive plant. There are now many works, situated principally in the Western States and Territories, successfully working arrastras, which are giving a high yield. A very large number of such works have been erected since 1876. Some few antedate that time. The Accidental

mine, for example, in 1863 and 1864, treated in the arrastra 1000 tons of ore yielding \$25 to \$75 to the ton. The Sexton mine in the same year treated ore which averaged \$37, and three tons of it gave \$191 to the ton. Many of these works have been built as a makeshift to take advantage of some peculiar conditions which were known at the outset to be only of a temporary nature. Some few of them have been altered from abandoned pan-mills, but most of them have been constructed especially to treat the ores of small but productive mines. Occasionally they are built in districts where only small mines are found, as custom-mills, and without any mining property attached to them. Others have been built to treat the tails of old mills, which can often be done with large profit, on even poor tails, when free water power can be had. When steam power has to be used, the tails must be correspondingly rich enough to pay for the expense of the fuel used to get the power. As the life of such works is dependent on circumstances which are almost accidental, it is usually very short, for which reason they are rarely built by companies with much capital; but they serve very useful purposes at small mines, and for working tails where the means at the disposal of the miner are very limited. They are employed to a large extent in working free-milling gold ores; also for treating silver ores containing gold which produces a doré bullion; but are not generally used on high-grade ores, as the pan is much quicker for such. Very little ore, averaging as low as \$10 (2¢) a ton, is treated in them. Usually the yield must be much higher.

The pavement should be made of the hardest and toughest rock that can be found in the country, preferably coarse rather than fine-grained, so as to remain rough after long use, not becoming polished by the grinding action, as this would tend to allow the ore to slip around rather than to be acted upon. It should also have the power of rubbing the fine particles of metal bright, so that they will be quickly attacked by the mercury. The rock used is generally granite or basalt, or some kind of compact quartz. New pavement is made at least 12 in. thick. The stone drags now used weigh generally from 200 to 1000 lb., or even 2000 lb. and are made of the same material as the pavement. The speed varies from 4 to 18 turns per minute. When power is used, it will average from 10 to 14 turns per minute. When the arms of a 10 ft. arrastra make 14 turns a minute, the outer drag has a velocity of about 400 ft. per minute. The average, however, will not be higher than from 200 to 300 ft. per minute. In the sides are either a series of round holes closed by wooden plugs, like those in the common settler, or a gate is provided on one side.

The difficulty of getting stone for the pavement, and of properly constructing it, has led to the invention of Paul's iron arrastra, which is

really a double pan. It varies from 6 ft. to 8 ft. 6 in. in diameter. The central portion of this machine is a pan, the muller of which is used for grinding. On the outside of this is a circular space 14 in. wide and 5 in. below the muller of the pan, in which a set of drags travel. It is fitted with a revolving amalgamated copper circle, which acts like the apron of gold-mills. It is run by power from below, like the pan. Such an arrastra, 8½ ft. in diameter, can treat from 7 to 10 tons of ore in 24 hours, according to the fineness and hardness of the ore treated. The whole machine weighs about 4½ tons, and requires 5 H.P. to move it.

The ore is first broken to a suitable size. When stamps are used for this purpose, the mortars are set without screens, and the ore is crushed dry, and delivered by the blow of the stamp, so that it varies from dust to the size of a pigeon's egg. A battery of 5 stamps will crush, in 2 hours, all the ore that a single arrastra can treat in 24 hours; such a battery, therefore, will be capable of feeding 12 arrastras. Charges now reach 600 to 1000 lb. The duration of grinding is fixed by testing the fineness of the pulp between thumb and finger. In some mills, chemicals are added as soon as the drags are in movement; in others, with the mercury. The chemicals used with gold ores are a little potassium cyanide to enliven the mercury, or some wood ashes or lye to neutralise the effect of any grease which may have got in with the ore. When the ores carry large amounts of silver, copper sulphate (bluestone) and salt, put in solid, are used. When the pulp has been ground long enough, mercury is added, in varying amount, generally from 100 to 150 lb.; but with high grade ores as much as 200 to 250 lb.

The working capacity of the simplest arrastra varies from one to two tons a day. A 12-ft. arrastra driven by power, with heavy drags, making 15 revolutions per minute, may treat two charges of 2 tons of ordinary ore in 24 hours, if very close work is not necessary; but this is the extreme limit of its capacity. The labour required is extremely small. One man per shift can easily take care of two arrastras. Sometimes when working on tailings, and running by water power, the disposition of the mill is such that the only labour required is for feeding and discharging, being much less than the work of one man. The owner of the mill may do all the work himself. With continuous action, the labour is still smaller, being simply that needed for repairs. Where water power is available, small overshot wheels, or turbine "hurdy-gurdies," are used. A simple arrastra for mule power can be built for 20% to 30%. The cost of 26 complete mills, having 92 arrastras, in 1880, was \$137,590, or an average of \$5292 for each works, or \$1495 (say 75%) per arrastra, including all the rest of the plant. This is considerably above the cost of ordinary arrastra mills. The arrastra at Scales and Wagner's works, Owyhee Co., Idaho (one of the best), produced in 1880

about 10,000%. The charge for treating custom ore in lots of 100 tons and over, is \$15, for lots of 50 to 100 tons, \$16, and a corresponding increase for smaller lots. The plant consists of a battery of 10 light stamps, 2 arrastras, 3 half-ton Wheeler pans, 2 settlers, and a retort. The working figures in summer, on a 12 hours' shift, were as follows:—

		\$	£	s.	d.
4 Men inside the mill at \$4.00 per 12 hours' shift	16.00	3	4	0	
2 Labourers in the summer, on ten hours' shift outwork, at \$3.50	7.00	1	8	0	
2 Engineers in winter, on 12 hours' shift at \$5.00	10.00	2	0	0	
Number of days' work, 330	..	..	..	..	..
Hours of labour to treat 1 ton of ore, 12.7	..	..	..	..	..
Cost of supplies per ton of ore treated	3.95	0	16	0	
Tons of ore treated in 1880 (tails about 300 tons), 1772.25 ..	..	..	..	..	..
Lowest yield of ore	38.00	7	12	0	
Highest yield of ore	600.00	120	0	0	
Average	115.85	23	3	6	
Yield of the tails	9.50	1	18	0	

The following table gives the total amount of supplies consumed, and the amount per ton of ore treated :

Items.	Total Consumption.		Consumption per ton of ore treated.		
	Amount.	Cost.	Amount.	Cost.	
		\$	£	s.	d.
Red fir cords	575	4,168 75	0.32	2	35 9 5
Quicksilver lb.	3,000	1,440 00	1.69	81	3 4
Salt do.	14,000	420 00	7.89	24	1 0
Bluestone do.	3,000	750 00	1.69	42	1 9
Lard oil gallons	20	40 00	0.01	2	0 1
Chemicals and sundries	..	200 00	..	11	0 5
Total	..	7,018 75	..	3 95	16 0

Atwood's Amalgamator.—The Atwood amalgamator, Figs. 196 to 198, essentially consists of two semicircular wooden troughs *a*, 19 in. long, 14 in. wide, and 6 in. deep, contained in a box, 4 ft. long, 21 in. wide, and 14 in. deep outside, set on a slope of 3 in. to the foot, so that one is 6 in. below the other, and each holding 350–400 lb. of mercury, above which revolve 2 wooden cylinders *b*, 18 in. long and 8 in. thick, studded all over, at regular intervals, with small iron pins set at right angles, whose slightly curved outer ends just clear the mercury. The troughs are separated by a riffle, and the cylinders are rotated by belts and pulleys *c f*, so that they revolve 60 times a minute. To ensure an even flow from the feeding-box *p*, an automatic contrivance is arranged as follows. The arm *g*, receiving motion from a wheel *f*, is attached to an iron lever *h*, bent at right angles and hinged at the angle, and having 3 holes in the end where *g* com-

municates with it, to permit the movement to be adjusted. A similar 3 holes are provided at the other end of *h* for the same purpose. The ratchet *i*, moving on the ratchet wheel *k*, which moves forward one ratchet for each rotation of the wheel *c*, is attached to a hole in *h*. An iron bar *l*, screw-threaded from end to end, is keyed on *k*, and serves as an axis of rotation; the under side of *l* is in contact with a half-nut *m*, kept in place by a counterpoise *n*, the lever *r* of which is attached to the

FIG. 196

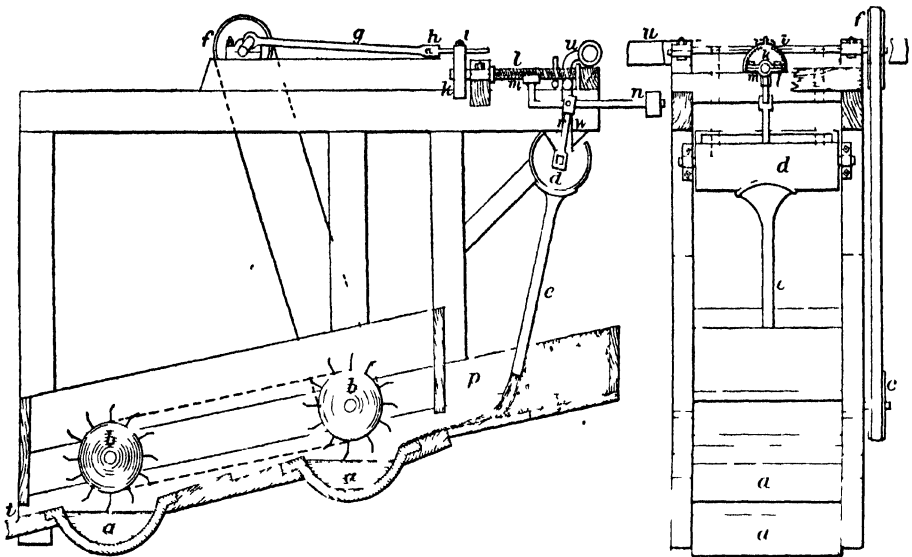
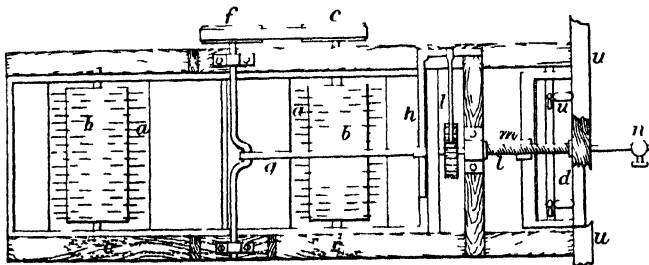


FIG. 197

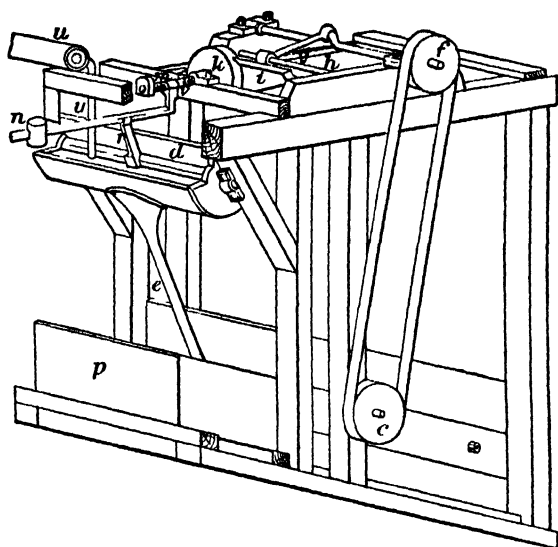


ATTWOOD'S AMALGAMATOR.

feed-box *d*. The feed-pipe *e* can be placed in any desired position by raising *n*. The ratchet and screw are so adjusted as to exactly wash the pulp out of the feed-box. The matters are charged into the rear of the feed-box *p*, which measures $3\frac{1}{4}$ ft. long, 18 in. wide, and 10 in. deep. The pipe *e*, which moves to and fro, is so set that the water, heated to 120° —

130° F., fed by the pipes *u*, will flow over the ore and wash it out into the amalgamator, through the first trough, over the riffles, through the second trough, and out at the discharge end *t*.

FIG. 198.

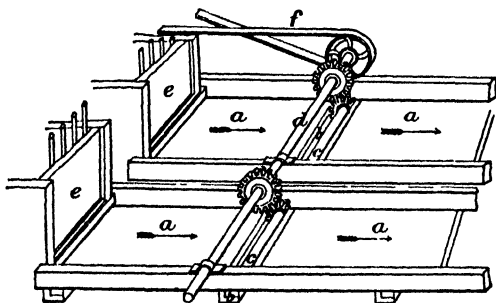


ATWOOD'S AMALGAMATOR.

Barry's Amalgamator.—The advantages claimed for Barry's patent gold-saving amalgamator (Fig. 199) are that it will save 5 dwt. more gold than the ordinary tables, that "sickening" is entirely avoided, and that the loss of mercury is smaller than with any other process.

a are ordinary copper tables, separated at *b* by a mercury trough or well, in which a copper tube or bar *c*, with an amalgamable surface, is set, with a small portion of its surface above the mercury, and revolving slowly against the flow of the auriferous pulp. The mercury surface of the roller

FIG. 199.



BARRY'S AMALGAMATOR.

should attract the fine gold, and carry it back into the well, where it will deposit itself so soon as the accumulation has become sufficiently heavy. By this process a fresh surface of mercury is constantly presented to the flowing gold, and thus "sickening" should be avoided. The tube *c* is

rotated by means of a shaft and toothed wheels *d*, driven by a pulley *f*; the plates lie adjacent to the stamper boxes *e*.

Berdan's pan.—Berdan's was one of the earliest and crudest kinds of pan, and is now almost obsolete. Its operation is very slow, only treating about 10 cwt. per 24 hours. It is occasionally employed on tailings.

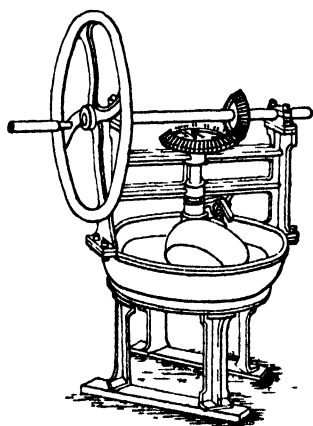
Boss Pan.—The boss pan, recently introduced in California, possesses no very remarkable feature to distinguish it from the many other existent forms. It is driven by gear-wheels; and the steam chamber extends up into the cone.

Britten's pan.—This is one of the simplest forms of pan, consisting only of a concave basin containing a muller, which is rotated by means of a manual fly-wheel and a pair of bevel-wheels, as shown in Fig. 200.

Chilian mill.—The Chilian mill is nothing more nor less than an edge-runner mortar-mill.

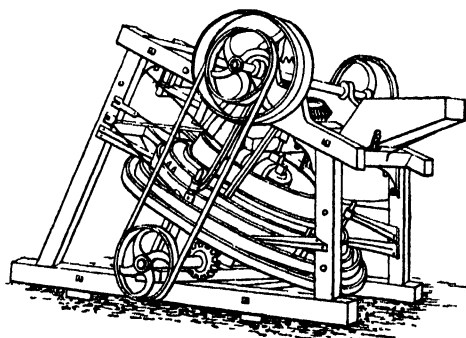
Denny & Roberts's pan.—The pan made by Denny & Roberts, Ballarat, Victoria, illustrated in Fig. 201, is highly esteemed in Australia, extracting 90 to 95 per cent. of the gold from ordinary tailings, and 70 per cent. from concentrated pyrites. The top pan is keyed on to the main inclined spindle, which makes 25 revolutions per minute when

FIG. 200.



BRITTEN'S PAN

FIG. 201.



DENNY AND ROBERTS'S PAN.

treating ordinary poor tailings, and 20 to 22 revolutions for rich blanketings. The middle pan runs about $\frac{2}{3}$ of the speed of the top pan, and is driven by a belt from the horizontal shaft, which gives motion to a small horizontal spindle, running in bearings bolted to the bottom frame of the machine. This spindle has a pinion keyed on to its inner end, which engages with a crown wheel that gives motion to the pan. The middle

pan has no connection with the main inclined spindle ; it is supported by a cast-iron bracket, on which it revolves the opposite way to the top pan.

The lower pan makes a little more than $\frac{1}{3}$ rd of the revolutions of the top pan ; it is also driven by a belt from the horizontal shaft, giving motion to a horizontal spindle running in bearings bolted to the bottom frame of the machine, having a pinion keyed on to its inner end, which engages with a crown-wheel for giving the pan motion in the same manner as the gear for the middle pan, only on the opposite side of the machine.

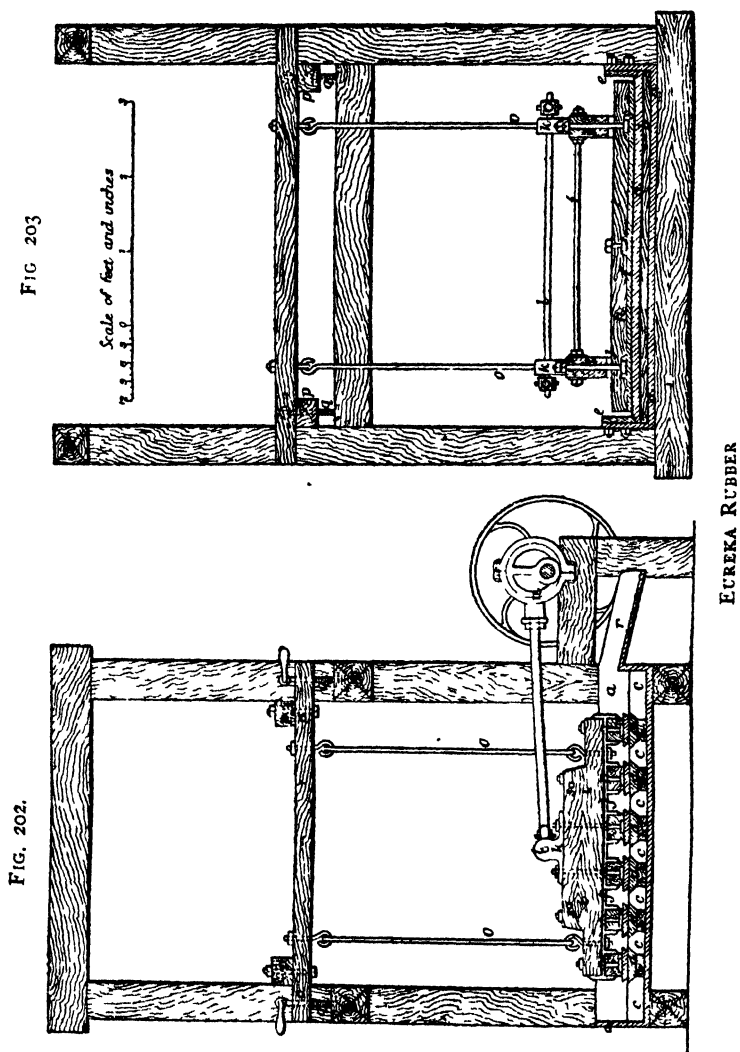
The top pan has a rotary grinder, the section of which is made to correspond with the section of the inclined pan, both having true conical lines, thus ensuring perfect grinding between the two surfaces without causing much wear and tear of parts in contact. This rotary grinder weighs about 7 cwt. ; its spindle runs in loose boxes ; these latter are kept in place by steel springs, which are made to act as levers for weighting the roller, by means of wheel-nuts. The operator may increase the crushing power of the rotary grinder 50 per cent. by taking a few turns on the wheel nuts ; this is only necessary when crushing the rough stuff left in the stamper-boxes on washing up, or any like material.

In addition to the rotary grinder, there are two disintegrators fitted opposite the main inclined shaft on the down side (the reader will understand that the down side means that portion of the pan which when revolving has passed the rotary grinder and is going to the lower side) ; these disintegrators weigh about 1 cwt. each, and are fitted with a white-metal shoe, which has a section corresponding with that of the pan. This shoe when fitted into the disintegrator presents a surface to the pan it stands in of only $\frac{1}{4}$ in. ; when the shoe is in its place, the down side of it is perpendicular to the surface of the pan it works in, and the upper surface forms an angle of about 45° with the surface of the pan, thus leaving a wedge opening of 45° for the material under treatment to pass. In order to ensure the passing of the material under the disintegrators, and to prevent its floating round the sides where it would escape the disintegrating action, the shoes are made with a fluted or corrugated surface, which admits the material under operation, thus preventing any escape. The material to be operated on is fed into the top pan, between the rotary grinder and the disintegrator ; then whatever gold is liberated by the frictional action of the disintegrator is carried by the revolutions of the pan directly to the mass of mercury in the lower angle of the pan, which presents a broad surface to take it up and amalgamate it, while the coarse matter is carried on and raised by the revolution of the pan, mixed with particles of mercury, and is

passed beneath the rotary grinder, which reduces it to fine powder preliminary to a successive action of the disintegrator and a successive action of the mass of mercury. As there is a continual supply of water to the upper pan, it overflows at its lower side, and the overflow, which consists of water and particles of matter fine enough or light enough to be carried with it, passes to the mercury amalgamating-well, which receives the liberated gold and silver. The particles of metal, by their specific gravity, assist greatly in amalgamating with the mercury in the well, as in falling from the edge of the pan they, having a perpendicular fall of 10 in., plunge into the mercury, where they are secured. From the mercury-well in the top pan, the pulverised matter and water pass on to a cast-iron apron, which directs them to the inside of the second pan, where they meet the upward current caused by the pan revolving, and are then carried into the annular channel, which contains mercury, when they pass round by the revolutions of the pan, and meet with the same action from this disintegrator and mercury as in the top pan. This second pan is fitted with 3 disintegrators, the same shape as those in the top pan, but lighter. The particles of fine metal, still associated with sand, pyrites, &c., that flow out of this pan, fall on to a cast-iron apron, which directs them into the upward current of the lower pan, when they are subjected to the same treatment as in the second pan. The top pan, from its fast speed, only liberates the coarse metal; the second pan, which revolves on a cast-iron bracket and has no connection with the main inclined spindle that carries the top pan, liberates the next heaviest of the remaining metal; and the lower pan, being large and having 4 disintegrators, besides a mercury-well, by its slow speed causes scarcely any agitation in the water, so that the very finest particles of metal are liberated and amalgamated. The disintegrators in the lower pan are of the same shape and weight as those in the middle pan, and the pan is supported by the same brackets as the middle pan, only it revolves on a different surface or part of the bracket; like the middle pan, it has no connection with the main inclined spindle; it may thus be made to run any number of revolutions by changing the pulleys.

Dickson's amalgamator.—This apparatus has a supply-hopper and a series of downward and upward passages connecting with scroll-shaped chambers, arranged so that the pulp from the stamp-mill is spread out in thin sheets, and the current is made to revolve with great velocity, so as to bring the gold and mercury in the chambers into intimate contact. The amalgam remains in the chambers, but the lighter particles escape from one chamber to another, and are finally allowed to pass away through the discharge-sluiice. The velocity of the water is regulated by plugs in the side of the discharge-sluiice.

Eureka rubber.—The Eureka rubber is intended to apply a certain amount of friction to the gold grains, by which rusty gold is cleaned and brightened, and attached slime is removed, thus facilitating the process of amalgamation. The machine, which is shown in Figs. 202, 203, 204,



and 205, consists of a cast-iron box *a*, measuring 56 in. square, 7 in. deep, and $1\frac{1}{2}$ in. thick, fitted with a false bottom. This latter, having a thickness of $2\frac{1}{2}$ in., is composed of three separate portions: a wooden die bed-plate *b*, extending the whole width of the box, $3\frac{1}{2}$ in. wide and 1 in. thick, passing between soft wooden dies *c*, measuring 11 in. by 4 in., and

cut transversely of the grain, and superposed by cast-iron dies *d*, 2 in. thick and made with flaring sides, the width being 4 in. at top, and only $3\frac{1}{2}$ in. at bottom, the object of the taper being to retain the wooden dies *c* in place, despite the wearing of the surface. Thus the floor of the box *a* is composed of alternating strips of iron and wood, secured from shifting

FIG. 204

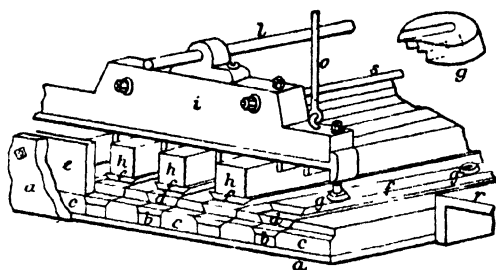
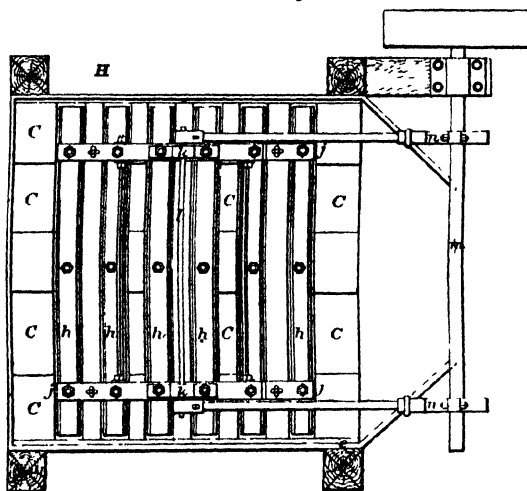


FIG. 205.



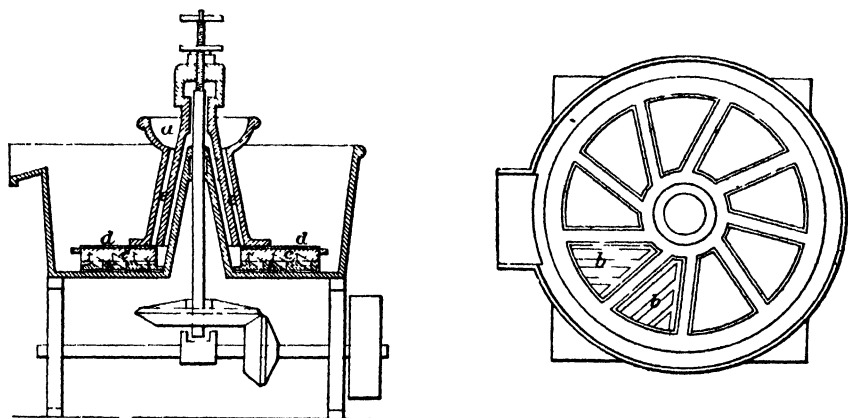
EUREKA RUBBER.

by means of strips of wood *e*, 1 in. thick, which are inserted $\frac{3}{4}$ in. deep into the dies and fastened by bolts to the sides of the box. Suspended over the false bottom is a wooden frame *i*, held together by bolts *s*, supported by the movable cross-piece *p*, through $\frac{1}{2}$ in. iron rods *o*, and made to move to and fro by means of an eccentric *n*. The bottom of the frame *i* carries a number of cast-iron foot-plates *j*, which, by the aid of flanges projecting $1\frac{1}{2}$ in., hold wooden shoes *h*, measuring $3\frac{1}{2}$ in. wide and $2\frac{1}{2}$ in. thick. These wooden shoes, again, are themselves shod with cast-iron plates *f*, which are 1 in. thick and tapering from 4 in. wide at

top to only $3\frac{1}{2}$ in. at bottom, sockets *g*, cast on the iron plates *f*, serving to attach them. Adjustment of the movable cross piece *p* is attained by means of a screw *q*, which permits the frame *i* to be set at any required interval from the floor. The shaft *l*, passing across the top of the suspended frame, is actuated by an eccentric which makes 55 revolutions a minute, and creates a sliding to and fro motion covering 4 in. The pulp is admitted to the rubber by a trough, and passes between the floor of the rubber and the suspended frame, the iron (*f*) and wooden (*h*) shoes of which are covered with amalgamated copper plates, that arrest the brightened particles of gold as they present themselves.

Fulton and Meekison's grinding and amalgamating pan.—The well-known difficulty of saving fine gold when in a current of water, especially when the material is fed on the surface, has led to many attempts being made to conduct the material down to the bottom of the operating machine. To effect this, Messrs. R. Fulton, C.E., Melbourne, and E. R. Meekison, travelling inspector of mining machinery, have patented an appliance, the novelty of which consists in the material to be pulverised being fed into the circular basin near the centre and top of the pan, and conducted down to near the bottom and below the top dies or mullers.

FIG. 206.



FULTON AND MEELKISON S GRINDING AND AMALGAMATING PAN.

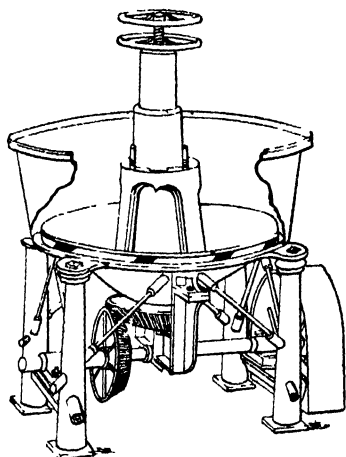
They also claim as a novelty the shape of the false bottoms and upper dies, by which the sand, after being delivered near the bottom and centre of the pan, is pulverised and discharged by centrifugal force at the outside diameter of the muller. Fig. 206 shows a section of the pan, dies, bottom, and the centre cone with circular basin or annular space (*a*) into which the material to be treated is fed; *b*, bottom dies; *c*, muller; *d*, plate carrying the muller; *e*, openings through which the material to be treated is conveyed to the bottom of the inside circle of the pan.

Gauthier's Shaking Amalgam Table.—This apparatus is designed to supersede the "apron" of amalgamated copper plate now used in most American gold mills. It consists of a silvered copper-plated table, as wide as the discharge of the mortar, and 16 ft. long, mounted on vibrating standards precisely as the Triumph concentrator, and having a similar "end shake" or longitudinal oscillation imparted to it by means of a shaft and eccentric below. The amplitude of the vibrations is adjustable, but is usually set at $1\frac{1}{4}$ in. This table, which occupies the position of the ordinary apron, receives the pulp from the mortar, and has a slight inclination downward from the battery, much less, however, than is necessary in a fixed apron, because the shaking keeps the sand in a loose and open condition independently of the current water. For the same reason, also, less water is required than with the usual arrangements. In a first experiment this plated table was mounted on the top of a Triumph concentrator, receiving the pulp after it had passed over the usual apron and plated sluices, and it was found that 7 per cent. of the total gold was saved on the table. The load proving too heavy for the Triumph machine to carry, the shaking table was mounted separately, so that direct comparison of results with those obtained from the apron and sluices of the other battery might be reached. The value of the shaking amalgam table is further confirmed by the experience made in the North Star Mill, where one of them, placed between the plated sluices and the concentrators, saves \$50 worth of amalgam per day, as stated by J. H. Hammond, E.M.

Hepburn & Peterson's pan.—The bottom of this pan, Fig. 207, has the form of an inverted cone, inclining toward the centre. The bottom is covered with 4 dies of corresponding form, which are secured in a similar manner to Wheeler's. Steam is introduced directly, without a steam-chamber in the bottom. In the centre of the pan rises a hollow pillar through which passes the driving-shaft. The shape of the muller corresponds with that of the bottom; at the centre it has an upright hollow cone, by which means it is connected with the hub or driver. The under side of the muller is furnished with shoes, between which, when attached to the muller, is a channel or radial passage left for the circulation of the pulp. The muller also contains radial grooves between the shoes, so that, when the latter wear down, the channel may still be large enough to permit an easy movement of the material. The muller is raised or lowered by means of a screw and movable nut at the top of the hub, the screw resting on the top of the driving-shaft to which the hub is keyed. The circulation of the pulp in this pan is effected without the use of wings or guides, such as are commonly employed in other pans for this purpose. When the muller is in motion, the pulp, passing between the grinding surfaces from the centre to the circumference of

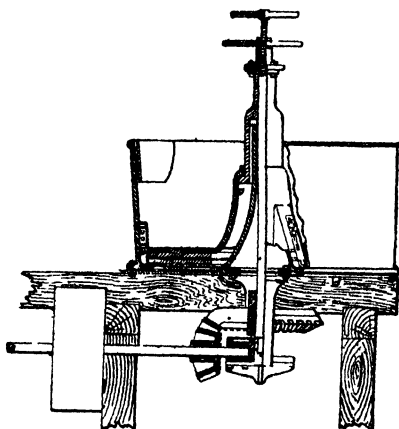
the pan, descends again by its own weight towards the centre, on the upper side of the muller : a movement promoted by the conical shape of the muller-plate. In the use of guide-plates or wings to aid the circulation, a difficulty is sometimes experienced in the tendency of coarser sand to settle and pack firmly, if the pan is stopped for a little while, giving much trouble in starting again. By thus dispensing with the use of wings, some inconvenience is avoided. The charge of the pan is about 1500 lb., usually working 4 hours on a charge. It runs at 60 to 70 revolutions a minute.

FIG. 207.



HEPBURN & PETERSON'S PAN.

FIG. 208.



HORN'S PAN.

Horn's pan.—Horn's pan, Fig. 208, is cast in one piece, and has a slightly flaring or irregularly concave surface. Around the dies is a depressed annular space 3 in. wide, traversed, as the muller rotates, by an arm which reaches to the bottom. The muller is raised by a screw at the top. The bottom is double, to afford an annular space for steam to heat the charge.

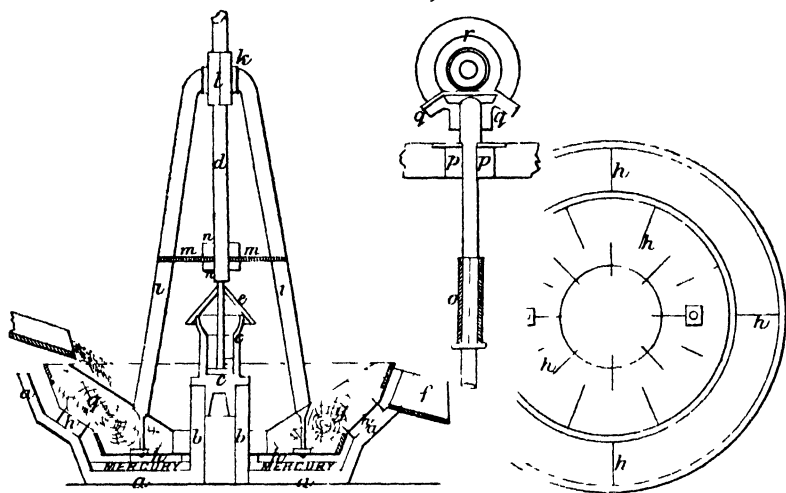
Hungarian bowl.—The Hungarian bowl or Tyrolcse mill, Fig. 209, consists of 3 main parts—the basin or bowl, the runner which revolves inside it, and the arrangement for transmission of power to the mill.

The basin *a* is of cast iron, $\frac{1}{2}$ in. thick, $6\frac{1}{2}$ in. high, 24 in. diameter at top and 18 in. at the flat bottom inside. An iron cylinder or pipe *b*, which rises for 4 or 5 in., is cast to the centre of the basin. This pipe has 3-in. outer, and $1\frac{1}{2}$ -in. inner diameter, and serves for the reception of the lower pivot bearings *c* of the spindle *d*. These bearings *c* consist generally of iron, and are protected against the influx of sand, &c., by a mantle or hood of sheet iron *e*. About $3\frac{1}{2}$ in. above the bottom is a hole

4 in. broad, in the rim of the basin, to which a tin or sheet-iron outlet *f* is riveted. The fixing of the basin to the planks of the floor is effected in a simple manner, either by screws through two ears, cast opposite to each other on to its bottom, or the latter is cast with a projecting rim, over which iron claws or hooks are driven into the floor.

The runner *g* is constructed of pine-wood, of the exact shape of the inside of the basin, but of somewhat smaller size, and is furnished at top with a wide funnel-shaped cavity, which communicates with a cylindrical hole of 5 in. diameter through the centre. Its size and shape are such

FIG. 209.



HUNGARIAN BOWL, OR TYROLFSE MILL.

that, when suspended centrally in its proper position in the basin—the hole just mentioned allowing it to go freely over the cylinder above described—its surface is parallel to the inner surface of the basin, and leaves an open space, $\frac{3}{4}$ in. broad round the side, and $1\frac{1}{2}$ in. at the bottom, and it projects about 1 in. above the top of the basin. Two thin sheet-iron hoops round the circumference protect it against cracking, and its bottom is armed with about 20 blades or knife-like scrapers *h*, of sheet iron, $\frac{1}{2}$ in. thick and $2\frac{1}{2}$ in. long, which are radially driven into the wood, each projecting exactly $\frac{3}{4}$ in. The central connection of the runner with the spindle is in some cases effected after the old method, by means of a tripod; but this will be gradually superseded by the generally applied and more practical new arrangement of a wrought-iron fork *i* with two prongs, which are driven through the runner and screwed tight, whilst a square collar *k*, formed at their junction, is slid over a square portion *l* of the spindle; 6 or 7 in. below this part, the prongs are con-

nected by an iron cross-bar *m*, which fits with a flat, round collar over the spindle, where the latter has, for several inches, screw turnings. Two nuts *n*, one above, the other below the collar, serve both for fixing the runner and to adjust it—i. e. to raise or lower it according to requirement. Each spindle has, a few inches above the upper collar, a coupling *o* for the purpose of throwing the runner out of action, without interfering with those of the other mills.

For the transmission of motive power from the axle of the water-wheel to the set of mills attached to a stamping-machine, three different methods are in use, viz. by endless straps and pulleys, by cranks, and by mitre-wheels. The last arrangement is the one most approved, and will therefore gradually replace the others ; for it ensures not only a better and more undisturbed working of the mills, but, though more expensive in the first instance, is ultimately cheaper as regards wear and tear. The construction is as follows : The pivot at the upper extremity of each spindle turning in a collar of white beechwood *p*, inserted in the horizontal beam of the framing, carries, at its end that projects a few inches above the beam, a small horizontal mitre-wheel *q*, 6 in. in diameter, with 21 teeth ; this wheel is turned smooth on its lower side, which rests upon an equally smooth iron plate, let into and secured by screws to the beam. Sometimes the end of the spindle is enlarged, and the rim rests upon the iron plate. All the mitre-wheels of the mills standing in one row are connected with corresponding vertical ones *r* of the same size, wedged over a horizontal iron axle that runs above the beam the whole length of the row of mills, and turns in white beechwood bearings at the points of support. According to the position of the two rows of mills, with reference to the axle of the water-wheel (for instance, whether parallel or at right-angles to it), mitre-wheels, the same size as those on the spindles, are fixed either at one end or on some part of the length of the horizontal axles. These are simultaneously turned, at the same time moving the runners of all the mills, by two other mitre-wheels, 10 in. in diameter, with 35 teeth, that are fixed on an iron axle, connected with the axle of the water-wheel, either direct or by intermittent gearing.

The relative diameters and numbers of teeth of the large and small mitre-wheels, viz. 10 : 6 and 35 : 21, are so calculated that if the water-wheel goes at its fullest suitable speed—i. e. at 15 revolutions per minute—the runners of the mills revolve 25 times during the same period. Their general speed is, however, only 16 to 20 revolutions per minute, and the motive-power required for one mill is but 0·04 H.P., or 25 mills can be turned by 1 H.P. When the runners of a set of mills are properly arranged, so that they revolve exactly vertical and central in the basins, each of the latter receives 50 lb. of mercury (the mills of some establishments receive only 30 lb., the reason for which will be given further on),

which forms a ring, 7 in. wide and $\frac{1}{2}$ in. thick, at the bottom, round the central pipe; and then a most important adjustment has to be made, namely, that of raising or lowering each runner, so that the lower edges of the iron blades, projecting from its bottom, stand just $\frac{1}{2}$ line above the level of the mercury. This distance, together with the breadth of the blades ($\frac{3}{4}$ in., as above given), has, by experience, been ascertained to ensure the greatest saving of gold and smallest loss of mercury. If once in proper working order, a set of mills requires afterwards but little supervision. Small tin gutters, attached to the main launder, distribute the crushed material equally to the mills, forming the first or upper row, and introduce it into the large funnel-shaped cavity of the runners, where it descends through the open space round the central pipe, and comes into contact with the mercury. It then undergoes much whirling about through the action of the iron blades—which, while promoting contact with the mercury, prevents deposition—and, gradually rising in the annular space between the runner and the inner wall of the basin, it passes out through the outlet spout into the mill straight in front, where the same process is repeated. The crushed material which passes a gold mill in this manner, from a battery of 3 heavy stampers, amounts in the average, per second, to 0.015 cub. ft., containing $\frac{1}{2}$ to $\frac{3}{4}$ oz. of sand and slimes.

On account of the ores being very poor in gold, and in order not to lose unnecessary time and money by too frequent interruptions of the process, the mills are allowed to run for 4 weeks at a stretch; the runners are then lifted out of the basins, and the whole of the mercury is removed. During this work, which generally lasts several hours, any small defects of the machinery are repaired, and all the different parts properly looked after. The amalgam is pressed through specially constructed double filters of chamois leather or strong close canvas, and the filtered portion, that still contains about $\frac{1}{8}$ oz. of gold per cwt., is returned to the mills, which are then set in action again. The mills of the lower rows are generally supplied with distilled mercury, for reasons which will be explained further on. The rather soft amalgam, obtained in the filters, is repeatedly pressed through close canvas, with hot water, till it becomes quite hard, when it is formed into balls of $3\frac{1}{2}$ to 4 oz. in weight, which are labelled and reserved for distillation. This amalgam contains about 25 to 33 per cent. of gold, and the relative quantities received from the upper and lower rows of mills are, on the average, in the proportion of 64.5 per cent. : 35.5 per cent.

When the mills at the end of a working period are opened, the greater part of the amalgam is always found collected round the central pipes of the basins. This fact led to the experiment of using smaller basins, in which the ring of mercury, instead of 7 in., was only 3 in. broad,

reducing the quantity of mercury to be used from 50 lb. to 26 or 28 lb. If this experiment had been generally successful, it would not only have lessened the outlay for mercury considerably, but would also in a great measure have reduced the loss of that metal; it was, however, found that from ores rich in galena—and they form by far the greater portion in Hungary—the produce of gold was 20 to 25 per cent. less than formerly. Only the ores of one mine—the Max shaft, on the Spitaler lode—which are comparatively poor in galena, yielded the same quantity of gold, whilst the loss of mercury was 50 per cent. less than previously, and they are therefore the only ones treated in small basins, supplied with but 30 lb. of mercury. The average loss of mercury sustained by the mill process is stated to be 20 oz. for 50 tons of ore rich in galena, and only $8\frac{1}{2}$ to 9 oz. for the same quantity of ore from the mine just mentioned. It is in reality, however, nearly 20 per cent. greater, for this amount is recovered during subsequent operations.

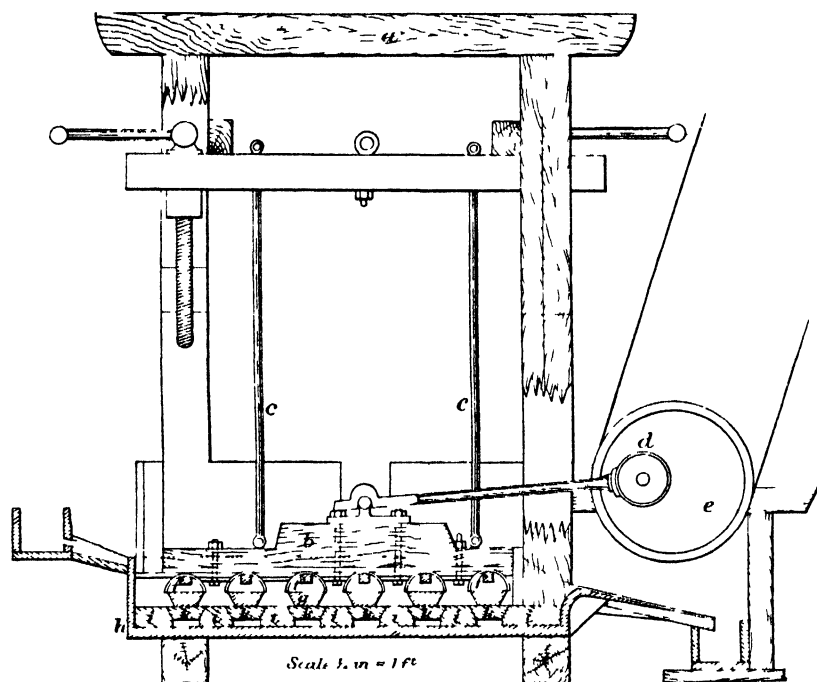
Another fact, experimentally discovered, touching the mercury and its use, as applied to the mill process, deserves mention: It is, that pure distilled mercury seizes upon gold eagerly up to a certain point—the limit of saturation—which is $3\frac{3}{4}$ oz. per 5 cwt.; beyond that point, when pasty amalgam commences to form, its action upon the gold is far less energetic. Taking advantage of this fact, by supplying the mills of both the upper and lower rows with pure mercury, and leaving it only in action till it had acquired the above limit of saturation, the yield of gold augmented 20 per cent., which, taking into account the smaller amount extracted by the subsequent operations, represented still 6·44 per cent. increase on the total produce of gold. This profit, considerable as it appears, was, however, much lessened by the expense and loss incident upon the repeated distilling of the mercury, its transport, and the more frequent interruptions of the process—the mills of the upper rows, for instance, having to be cleared every 5 to 6 days. This caused a less yearly produce, with a greater amount of manual labour, and therefore this method of amalgamation has been discontinued as far as regards the mills of the upper rows, but is still in operation for those of the lower ones, because the stuff travelling through them is so much poorer in gold that the limit of saturation of the mercury is only reached after every 28 to 30 days, when the mills of the upper rows have to be cleared also.

These pans are erected in two rows, the first $4\frac{1}{2}$ in. above the second in front of the stamps. The ore which has to pass the pans must be stamped very fine. It is generally reckoned that 2 pans are required for every ton of ore crushed.

Hunter's rubber.—This machine is illustrated in Fig. 210. It is rather complicated, combining, like most other American gold-saving appliances, two or three different actions, viz. grinding, amalgamation,

and concentration. It has a similar appearance to the old shaking-tables, and its motion is also similar; but in detail it differs materially from the former. From the well-stayed frame-work *a* depend two

FIG 210.

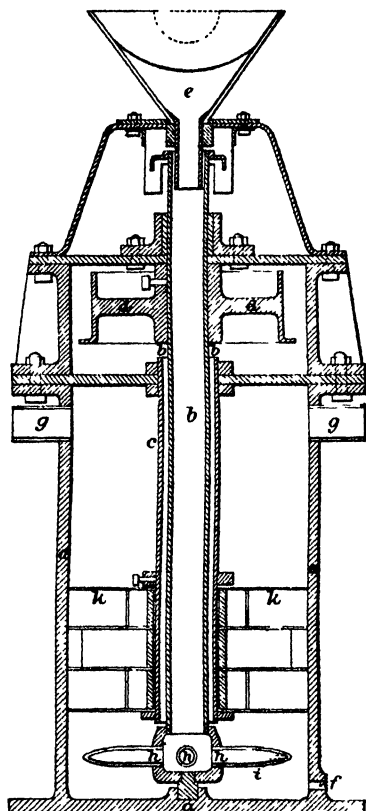


HUNTER'S RUBBER.

bearers *b*, by means of 4 bars of round iron *c*, and these are rocked 50 times a minute by 2 excentrics *d* and pulleys *e*, with a stroke of 5 to 7 in. Six pieces of pine-wood *f* are bolted to the bearers *b* longitudinally, their tops being round and their bottoms square, where they are armed at the bottom with the same number of shoes *g*, all these being the really movable parts of the machine. In a strong cast-iron box *h*, a false bottom is laid by means of alternate strips of wood *i*, and cast-iron dies *k*, in the same longitudinal fashion, so that the shoes rub upon the dies and thereby grind the ores. At the same time the tops of the wooden strips *f*, are covered with electro-copper plates, "and as they are immersed, any, in fact nearly all the float gold liberated by the grinding is collected at the apex of each cylindrical copper plate, and the pyrites are also concentrated in this box. This is a very valuable machine, as it collects from 10 to 12 per cent. of gold that would otherwise float away with the blue slimy water, which it is well known is allowed to escape elsewhere." (Thureau.)

The Huntington and Koch amalgamator (Fig. 211) consists of a cast-iron vessel or pan *a*, which has an outlet *f* at the bottom, for running off the amalgam or emptying the amalgamating metal from the pan; also several outlets *g* at the top, through which the tailings are discharged. Passing down the centre of the pan is a vertical pipe *b*, into which the ore or tailings are fed through the fixed hopper *e*. They are then discharged, or sucked, into the amalgamating metal from the pipe *b*, by means of the tubular arms or nozzle *h*, made of a tapering ovate form, with an elongated slit *i* on the hinder side relatively to the direction of rotation. These arms create a vacuum in passing through the amalgamating metal. A stationary pipe or sleeve *c* prevents the contact of the revolving pipe *b* and the amalgamating metal. Attached to the sleeve are vertical and horizontal screens *k*, to prevent the rotation of the amalgamating metal, and to ensure the mingling of ore or tailings with same. The pipe *b* is driven by a pulley *d*. In operating with this apparatus, the ore or tailings, after being crushed in a battery or other suitable appliances, are delivered continuously into the hopper *e*; they then descend the revolving pipe *b*, and are ejected from the slits *i* of the arms *h*, so that they are mingled with, and rise through, the amalgamating metal. The worked material, being discharged through the outlets *g*, flows thence to a pan containing rakes revolving at a slow speed; from this it is discharged to the tailing heaps. Either mercury when ore is crushed wet, or molten lead with dry pyrites, can be used as the amalgamating metal in this apparatus. On upwards of 100 tons of ore put through, in competition with ordinary appliances at Sandhurst, Victoria, the results were, according to the nature of the ores, from 11·33 to 18·2 per cent. more gold than from batteries with tables, ripples, blankets, &c. The cost of working does not exceed that of present appliances, and with a plant specially erected for its use the cost will be materially lessened. The machine can be connected with batteries,

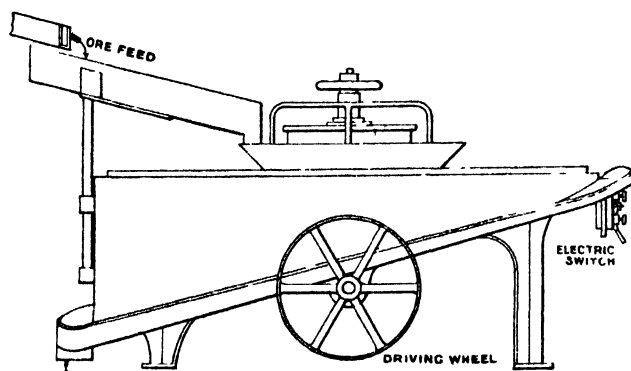
FIG. 211.



doing away with all tables, ripples, blankets, &c. With ores containing pyrites, a very large percentage of the gold therein is saved. But where the ore contains much of any mineral which has a tendency to flour the mercury, the machine incurs a great loss of mercury, unless the stuff is previously reduced to a very fine state in Chilian mills.

Hydrogen Amalgamator.—The method pursued with the hydrogen amalgamator, introduced by C. B. Molloy, and shown in Fig. 212, is (*a*) to maintain the “quickness” of the mercury, and (*b*) to ensure continual contact between each separate particle of the pulverised ore and the quick mercury. The apparatus for accomplishing this consists of a shallow pan about 1 in. in depth, and 41½ in. in diameter, which contains mercury about ½ in. in depth. In the centre of this pan is a porous jar, so placed and fixed that the mercury cannot enter or move it. Within this jar is a cylinder of lead and a solution of sulphate of soda. This lead cylinder, which constitutes the anode, is connected with the positive pole of a small dynamo machine, while the mercury is connected with the negative pole of the same dynamo. When the current passes, oxygen is evolved from the surface of the lead anode, while hydrogen is evolved from the surface of the mercury. This action, which is apparent to the eye, is of course due to the decomposition of the electrolyte formed by the solution of the sulphate of soda. The mercury combines

FIG. 212.



HYDROGEN AMALGAMATOR.

with a portion of the hydrogen and so forms a hydrogen-amalgam, while the excess of hydrogen so formed passes away. Now, while the mercury is thus charged with hydrogen it cannot oxidise, because of the presence of an excess of hydrogen. Thus, no matter what the character of the ore, the mercury, under these conditions, is always quick and greedily attacks and absorbs the gold into itself.

Now as to the means whereby the pulverised gold ore is brought

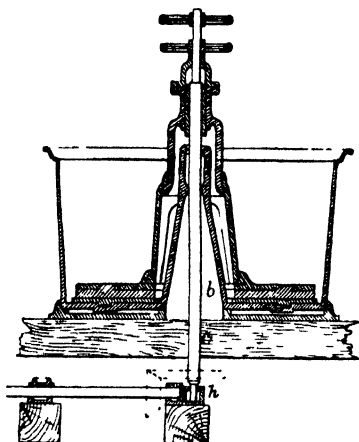
into absolute and maintained contact with the quick mercury. Floating upon the surface of the mercury is a disc 40 in. in diameter, which dimension leaves a narrow outside channel all round the edge of the pan where the mercury is uncovered. The centre of the disc has a circular hole in it, so as to clear the porous jar by about 2 in. This central opening in the disc has a rim about 2 in. high, which forms a hopper. The disc as it floats on the mercury is slowly revolved by simple mechanism. The pulverised ore as it leaves the stamps or other crusher flows into the hopper accompanied by a stream of water, and is then by centrifugal action carried under the revolving disc and rolled round in the mercury, in ever-increasing circles, until it reaches the periphery of the disc and consequently the outward channel between the edge of the disc and that of the pan. Here, freed from the pressure of the disc, the pulverised ore floats up and over the edge of the pan, and passes away, leaving behind it in the mercury every atom of amalgamable gold it previously contained. This perfect extraction is due to the rolling action which separates each particle of the ore and rolls it for some ten seconds in the bright, quick mercury. The whole machine weighs only some 5 cwt., and its capacity is 6 tons a day. Published results of the working of the process on a variety of sulphuretted and other ores show a yield ranging from $73\frac{1}{2}$ to $96\frac{1}{2}$ per cent. of the assay value, at a cost not exceeding 9d. a ton for treatment.

McCone's pan.—This pan, Fig. 213, is 5 ft. in diameter and 28 in. deep. It is flat-bottomed, and made either with or without a steam-chamber. When the latter is desired, the false bottom is cast separately, with a rim 1 in. deep, and is then bolted to the main pan-bottom, thus forming the chamber. There are no standards or legs for the pan to stand upon, the bottom being a square-cornered plate of iron, projecting beyond the pan-rim, and it may be bolted directly to the timbers on which it is to rest. The bottom, with its central hollow cone, may be cast in one piece with the pan-rim, or, instead of the latter, a simple flange may be cast, corresponding in size with the rim, to which flange the rim, which may then be either a cast piece or be made of sheets of iron riveted together, is bolted.

An improvement has lately been made to save the wear of the rim or side of the pan, and prolong its usefulness, by placing in the bottom of the pan a false rim or circular facing for the pan-side, about 9 in. deep. This is cast in segments, and made to correspond in form to the rim of the pan. When fixed in place, it saves the pan-rim from wear in that part which would otherwise suffer the greatest degree of friction, just as the shoes and dies protect the pan-bottom and muller-plate. When worn thin by the friction of the pulp, the plates may be removed and new ones substituted for them. The driving shaft or spindle *c*

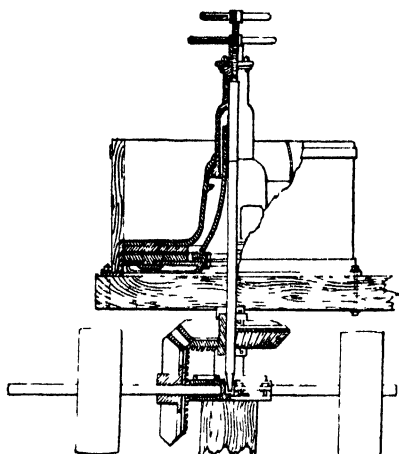
passes up from below through the central hollow cone *b*; but its point of support is usually independent of the pan, resting, in such case, in a step-box *k*, which is fixed on a timber below. Some, however, prefer to have hangers bolted to the bottom of the pan, and furnishing the support for the driving shaft, so that, if the foundations of the pan settle, the relative position of the several parts is more readily maintained. The step-box is cast in one piece, with a bearing for the end of the shaft on which the vertical mitre-wheel and pulley of the common driving gear are fixed.

FIG. 213



McCONNEL'S PAN.

FIG. 214.



PATTON'S PAN.

The driver or hub, which is secured to the vertical shaft, is in two parts, an upper and a lower. The upper is fixed to the shaft by two strong feathers or sliding keys. The base of the upper driver is cast with lugs, or projections, which fit into corresponding recesses in the top of the lower driver, by which means the latter is supported and set in motion. Above the upper driver is a cap-piece, carrying the usual screw and nut arrangement for raising and lowering the muller, the bottom of the screw resting on the upper end of the vertical shaft. The lower part of the driver has three or four stout lugs, or projections, at its base, which fit into carriers on the circular part of the muller. These carriers are also made to serve as the means of aiding the circulation of the pulp, as they assist in directing the current towards the centre when the muller is revolving. For this purpose, they are sometimes cast 5 or 6 in. high, presenting a curved surface, not shown in the case illustrated, to the pulp, and forcing it towards the centre of the pan. By this means, the guide-plates or wings, usually fixed to the side of the pan, but which to some extent obstruct the motion of the pulp, are dispensed with.

Grooves for attaching guide-plates, are, however, cast in the pan-rim, so that those who prefer may use them. The dies and shoes used in this pan resemble, in many respects, those of other pans. There is $1\frac{1}{2}$ in. space between the outer edge of the die and the edge of the pan, and a similar space between the adjacent edges of the dies. The shoes, between which are similar spaces, and which also have radial channels or grooves on their under side, to facilitate circulation, have the same radial width as the dies. The radial width of the muller-plate is a little less than that of the shoe and die, in order to allow a freer inlet and outlet to the pulp. The muller makes 60 to 80 revolutions per minute. The pan takes 4500 lb. of pulp at an ordinary charge, and sometimes more. It is set up very simply, being bolted to timber supports below, and is put in motion or arrested by the application or withdrawal of a tightener to the driving belt.

Manes's electric pulveriser and amalgamator, for saving rusty and fine gold and mercury, was originally intended for use in hydraulic washing. It is claimed that it will save from 50 to 75 per cent. of all the gold and mercury that passes through the machine, as the rusty gold will be perfectly scoured and electro-plated with mercury, and thoroughly amalgamated, by the rapid action of the electrical steel brushes and steel mullers that revolve inside of the series of steel cylinders in the machine, placed one above the other, and made cone shape, and connected with spouts. The large end of one cylinder is placed under the bottom of the small end of the next one, and so on, forming inclined planes for the sand or crushed ore to run down by its own gravity, which is assisted by streams of water and mercury, constantly fed into the machine from a hopper on the upper part of the machine; and the powerful current of electricity is constantly passing through the sands or ore as it goes from one cylinder to another. As it is thoroughly mixed at the same time with the mercury by the steel brushes, no gold should escape without having been amalgamated. The material passes through a movable iron spout into settling tanks, where the cleaning-up is done. The machine only stops when repairs are needed, tail-water is used for driving, and only one man is required for each machine. The apparatus can equally be used in stamp mills.

Otto's electric gold-saving apparatus has been tried in Australia with varying success. The apparatus is strong and simple, and relies upon the increased activity of mercury under galvanic influence. The zinc terminal of the battery is placed in contact with the mercury and insulated from the carbon poles, which are fixed opposite each other above and across the mercury. The water acts as a conductor for completing the circuit through the mercury. The result is said to be a constantly bright surface on the mercury, thus augmenting its

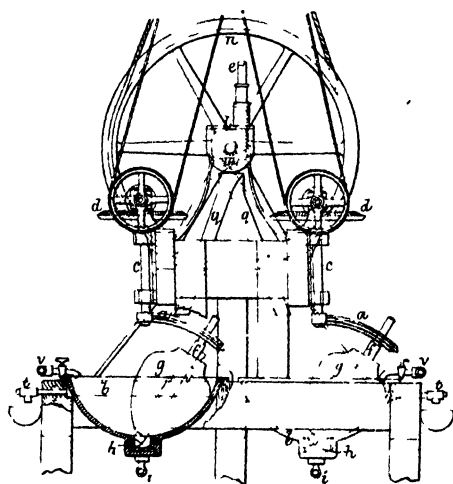
amalgamating power. The apparatus should immediately follow the stamp battery, dispensing with blankets and ripples.

Patton's pan.—In Patton's pan, Fig. 214, the wooden sides are vertical, and the staves are held by a strong iron hoop upon an iron flange or shoulder of the bed-plate, which rises inside the pan as high as the top of the muller. The bottom is cast in one piece, and is provided with a chamber beneath for the admission of steam to heat the pulp, and thereby promote amalgamation. The pan is 5 ft. in diameter and 2 ft. deep. The motion of the muller is communicated to it from below. The distance between the grinding-surfaces is regulated by a screw on the top of the vertical shaft.

Peck's amalgamator.—The novelty in Peck's amalgamator consists in having the pans made of copper and amalgamated prior to use, the pans being arranged in a descending series, and driven by a mechanical arrangement which produces a vibratory motion in addition to the usual rotary one.

Readwin's amalgamator.—In the amalgamator recently introduced by Thomas Alison Readwin, so long known in connection with the Welsh gold industry, a framing, Fig. 215, supports the pans *b*; *c* are

FIG. 215.



READWIN'S AMALGAMATOR.

vertical shafts driven by gearing *d* from the driving shaft, the gearing being disconnected when required by shifting the pinion out of gear along the shaft by means of levers. The shafts *c* carry each a hooked arm *a*, in which freely rests the axis or spindle *f* of the pestle *g*, so as to carry the pestle round, at the same time allowing it to rotate about its axis, the half vertical section of the pan approaching approximately the rounded surface of the pestle. The bearings of the vertical shafts *c* are formed of hard wood, prepared asbestos, or other material which will wear well when lubricated

with water. The lubricant is supplied to the bearings through holes formed in the bosses of the large gear wheels *d*. The pestle-axis *f*, which is made of hardened steel or phosphor-bronze, is fitted to the pestle-body so as to be easily removable, and so that the pestle-body can be shifted lengthwise of it at pleasure. The bottom of each pan

is recessed so as to receive a hardened steel or phosphor-bronze cup *h*, in which works the lower end of the pestle-axis, and which holds the mercury for amalgamating the gold. Eyed tap-screws *i* are screwed into the bottom of the pans to allow of the mercury or amalgam being run off. A wire or bar is passed through the eyes of the screws *i*, and secured by a lock and key, so as to prevent unauthorised withdrawal of the contents of any of the pans. A trough *l*, carried above the pans, has a screw-bladed shaft *m*, which revolves by means of the wheels *n*, driven from the shaft *o*, which screw feeds the crushed ore fed from the hopper to each of the pans by means of shoots or spouts *g*. The regulating tube *e* is capable of being shifted up or down, so as to regulate the supply of crushed ore to the trough *l*. Sliding shutters, fitted to the bottom of the trough, regulate the openings so as to allow the requisite quantity of ore to pass through in a given time. An overflow pipe carries off the superfluous ore as it collects at the farther end of the trough, whence it is returned to the feed-hopper. Overflow pipes *t* allow the surplus water, together with the baser metals and gangue in a finely-divided condition, to escape from the pans by troughs to receivers. Two or more outlets may be applied to each pan when required at different levels. Pipes *v* running the length of the pans supply water.

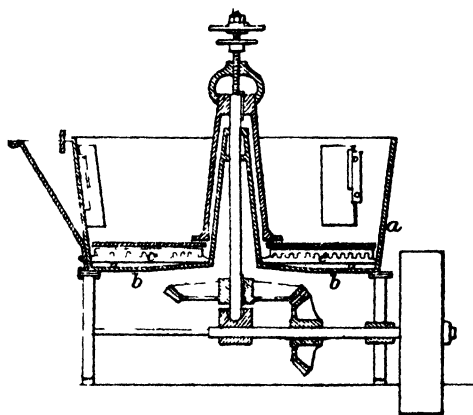
The operation of the machine is as follows:—The cups *h* are filled with mercury. Water is then run into the pans up to the level of the outlets *t*. The pestles are then set in motion by throwing the bevel-wheels *d* into gear. The hopper being filled with ore crushed to the required size, the ore falls into the trough *l* at such a rate as may be allowed by the adjustment of the shutters of the shoot *e*. The screw *m* being set in motion carries the ore along the trough, and causes it to pass in a uniform layer over the adjustable apertures, the screw being so connected with the shaft that it shall revolve at a proportionate rate with the pestles, supplying the necessary quantity of ore per minute to each of the pans. The ore will thus be supplied gradually and in the proper quantities to the pans, and by the crushing and grinding action of the pestles on the pans it will be reduced to a finely divided condition, in which state it is presented to the mercury in the cups *h*. The mercury catches the amalgamable portion of the precious metals contained in the pulverised ore, and thus forms the required amalgam. During the process, the water, together with the baser metals and gangue, is continually flowing through the overflow pipes *t*. When it is required to withdraw the mercury and amalgam, the wire rope or rod is withdrawn from the eyes of the tap-screws *i*, and the mercury and amalgam are run off as required.

Watson and Denny's pan.—The main body of this machine, Fig. 216,

weighing about 3 tons, is a wide circular iron casting *a*, about $5\frac{1}{2}$ ft. wide and $2\frac{1}{2}$ ft. deep, the bottom *b* of which is flat near the periphery, but rises slightly towards the centre. A false bottom *c*, consisting of corrugated segments, is placed in a horizontal position above the true bottom, the inner end resting on the upper part of the true bottom, while the outer end rests on supports. These segments have projections at both ends, which fit into places provided for their reception, to prevent them from shifting. There is an elongated wedge-shaped space left between the lower segments and the true bottom; this is partly filled with mercury when working, and it is here that the process of amalgamation takes place. The upper grinding segments are attached to the carrying plate, which communicates a rotary motion to them from the driver. Between these grinding surfaces the ore is ground to an almost impalpable state; and the currents formed by the rotary motion, the inflow and outflow of material, together with the effect of the guide blades, causes the reduced material to pass numerous times against and into the mercury before escaping into the classifier, prior to being conducted away. As the mercury is kept in motion by the produced currents, it is constantly presenting fresh surfaces to the material.

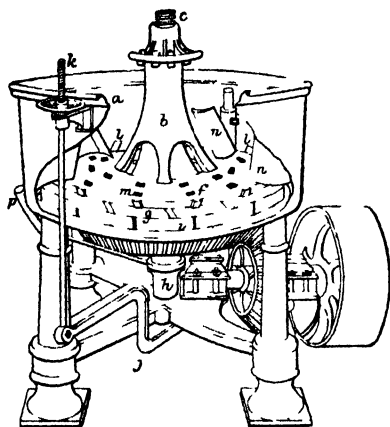
FIG. 216

I



WAISON AND DENNY'S PAN.

FIG. 217



WHEELER'S PAN.

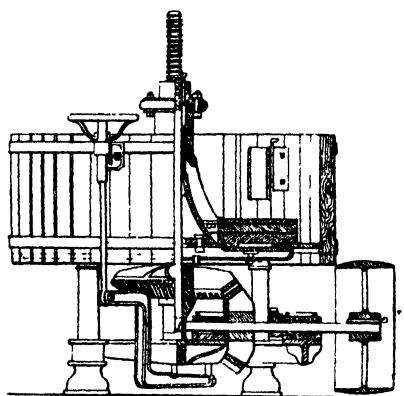
The classifier is fixed on the outside of the machine, its three outer sides tapering towards the bottom, where it communicates with the interior of the machine by a hole. In the classifier, a deflecting plate is so arranged that the size of the grains of tailings can be regulated to suit different circumstances. All grains over a certain size are automatically separated and collected in the classifier by this means, and settling by gravity in the comparatively still water, are sucked through the hole at

the bottom into the machine, and re-ground, the waste passing away with the overflow, and the possible loss of valuable material is thus reduced to a minimum. The gearing for driving purposes is placed under the machine, being thereby protected from dust ; and there is thus no danger of oil from the bearings getting into the machine to sicken the mercury. The machine is made by the Watson and Denny Gold and Silver Extracting Company, Melbourne.

Wheeler's pan.—The Wheeler pan of ordinary size, Fig. 217, is about 4 ft. in diameter at the bottom, and 2 ft., or a little more, in depth. *a* is the rim of the pan, in the centre of which is the hollow cone *b* rising from the bottom, with which it is cast in one piece. Through this cone passes the vertical shaft, which, being driven by the gearing below the pan, gives motion to the muller by means of a driver keyed to the shaft. The muller is provided, on its under side, with shoes *g* that form the upper grinding surface. They are attached to the muller by means of two lugs or projections *f*, which are received in corresponding apertures in the muller-plate, and securely wedged with pieces of wood. The lower grinding surface is formed by the dies *i*, which are usually 4 or 8 in number, covering the greater portion of the pan-bottom, and secured to it in a manner similar to that by which the shoes are fixed to the muller. There is a radial slot or space between the dies, which is commonly filled with hard wood. Below the bottom is a steam-chamber for heating the pulp. The vertical shaft or spindle rests in a step-box *h*, to which oil is conveyed by the pipe *p*. A vertical pin passes downward through the centre of the step-box, in contact with the shaft and resting its lower end on the lever *j*. This lever may be raised or lowered slightly by the hand-wheel on the rod *k*, thus raising the muller from the dies, if desired. The shaft is also furnished with a screw *c*, by means of which the muller may be raised up entirely above the rim of the pan for the purpose of cleaning up or of changing the shoes and dies. In order to impart an upward current or movement to the pulp, there are inclined ledges *l* on the rim of the pan, and smaller ledges *m* on the periphery of the muller, but inclined in the opposite direction. The pan is also provided with wings or guide-plates *n*, 4 in number, which serve to direct the moving pulp toward the centre. They are fitted into and may be removed at pleasure from a T projection on the pan-rim. The muller is usually caused to make about 60 revolutions a minute ; it requires 2½ to 3 H.P. Its ordinary charge is 800 to 1000 lb. In some mills, a still larger charge is worked. The capacity of the pan is sometimes augmented by adding a rim of sheet iron, so as to increase the height of the side. The treatment of the charge usually requires 4 hours. The shoes and dies wear out in 3 to 6 weeks, though they are made to last longer in some mills, their duration depending greatly upon the order in which the pan and all

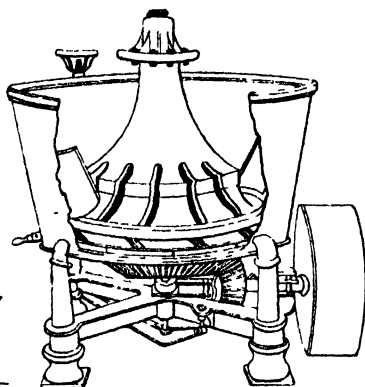
its principal working parts are kept. Neglect in oiling the working parts of the running gear is apt to cause unequal wear, the vertical shaft gets loose and out of line, the grinding-surfaces cease to work together evenly, and the efficiency of the pan is greatly impaired, while the costs of working are very much increased. Mill-men generally prefer a shoe and die of moderate rather than excessive hardness. The former wear out faster, but are thought to grind more efficiently. Such are usually cast of an equal mixture of white and soft iron.

FIG. 218.



WHEELER'S PAN.

FIG. 219.



WHEELER AND RANDALL'S PAN.

A more modern form of Wheeler's pan is shown in Fig. 218. It bears considerable resemblance to Patton's, already described (p. 512).

Wheeler and Randall's pan.—Fig. 219 presents a view of the "Excelsior" pan, devised by Wheeler and Randall. It differs from others chiefly in the form of the bottom, which is conoidal. The object of this device is to produce surfaces of such form as to ensure perfect uniformity of wear and the highest degree of grinding effect. Its efficiency, in this respect, is attested by the experience of practical mill-men. It is not, however, so generally used as the ordinary Wheeler.

The dies, muller, and shoes have a form corresponding to that of the pan-bottom. They are secured in place in much the same way as in the Wheeler pan. There are guide-plates to assist in directing the movement of the pulp, and openings in the muller between the shoes for its free passage between the grinding surfaces. The gearing of the pan, step-box, and driving-shaft, and means of raising the muller, do not differ materially from the common Wheeler pan. This pan is made of various sizes; the largest is $4\frac{1}{2}$ ft. in diameter, and treats 3000 lb. of ore at a single charge. It weighs 5000 lb.

Blanket-tables.—In Hungary, Victoria, and some parts of America,

the matters leaving the last row of pans or bowls fall into a narrow trough, from which, by means of wooden shutters, they are equally distributed over a number of inclined tables or "strakes" clothed with blanketing or canvas. When no pans or bowls are used, the tables immediately succeed the battery.

The Victorian form consists of a wooden floor, with varying length, and laid at a varying pitch or incline. It is fixed securely, and in such a manner as will enable the pitch to be altered, always observing the utmost regularity in its arrangement, and ensuring precisely the same degree of pitch throughout the entire length. The surface of the table is made perfectly smooth and true. By means of strips of wood $2\frac{1}{2}$ in. wide fastened to the floor, it is subdivided into "strakes," one for each stamp in the battery, when pans or bowls are not employed; these strips afford a place for the man to step on who changes the blankets. Sometimes the table is broken transversely into sections of 3 ft. or so, the upper edge of each section being about 2 in. below the overlapping edge of the one next above. The surface is covered throughout with closely-woven baize or blanket, laid on with extreme care, so as to lie flat and cling to the boards. The mixture of water, mercury, and disintegrated matters passes over the surface of the blanketing in a thin stream. The high specific gravity of the gold and pyrites causes them to descend to the lowest stratum of this stream, by which they are brought into contact with the fibres of the blanketing, and are induced to settle among them. From these they are subsequently dislodged by rolling up the blankets in turn and washing them in clean water.

The Hungarian apparatus differs in that it is made even throughout (without any steps), and that the covering medium is a very rough kind of canvas, resembling sackcloth, and specially prepared for the purpose. The length and breadth of the tables vary in different establishments from 6 to 12 ft. and from 13 to 18 in. respectively, with an inclination of 1 to $1\frac{1}{2}$ in. per ft.; they are constructed by preparing an even flooring of boards, laid crossways, of the required size and inclination. This is divided into equal spaces by longitudinal partitions, 2 to 3 in. broad and 2 in. high, with cheeks of similar dimensions fixed along the sides. The number of strakes constructed in connection with a certain number of mills is usually in the relation of 6:8, with one table extra. Thus 8 mills have 7 tables, 6 of which are in action at a time, whilst the 7th is opened when 1 of the 6 is closed for the purpose of washing the canvas pieces. These pieces are generally cut of such a size that 2, for instance, cover a table 6 ft. in length, including the overlap. Numerous experiments have proved this coarse canvas, partly as regards cheapness and partly in efficiency, to be better than either blanketing or two other peculiar methods of producing the required rough surface, viz. (1) by scratching the boards,

and (2) replacing the latter by finely corrugated iron plates. The wear and tear of the cloth, which is prepared of a width to suit that of the tables, is about 16 ft. for working 50 tons of matter, the pieces lasting generally 6 to 8 weeks. They are washed in tanks, in the same manner as the blanket pieces used in Victoria. The operation is performed by boys, every 1 to 3 hours, according as the supply of material is greater or smaller, one boy commonly sufficing to work 10 or 15 tables. The stuff collected in this manner in the washing-tanks forms 2 to 3 per cent. of the whole amount of ore crushed, and consists (in Hungary) of finely-divided galena, pyrites, and gangue, through which are distributed fine free gold and small globules of mercury, escaped from the mills.

Many conditions govern the success or failure of the blanket-table. (1) When the gold is stamped to an excessive degree of fineness, or is flattened into tiny thin plates, it becomes what is known as "float" gold, i.e., owing to the minuteness of the particles, or their flattened shape, other forces tend to counterbalance the effect of their great specific gravity, and instead of sinking as usual they actually float away. (2) The supply of water must be exactly adapted to the nature of the material under treatment: if too little, the material will be unevenly distributed, and will clog the blankets; if too great, it will wash away some of the gold. (3) Too rapid inclination of the tables is often a source of loss: it should scarcely ever be greater than 1 in 14 to 16. (4) The length of the table is of importance in reducing the inevitable loss to a minimum: for, though by far the largest proportion of the valuable matters is deposited on the first few feet of the strakes, yet, however far the latter may have been continued, they will always catch some particles. In practice, it would be inconvenient to much exceed 30 ft., but 20 ft. should be regarded as a minimum figure. (5) The interstices of the blanketing would, in course of time, become choked with heavy matters, and would then cease to be a receptacle. The washing and renewing must therefore be performed at sufficiently short intervals. The frequency with which this is repeated will much depend upon the character of the material, being increased when it is "slimy," and when much pyrites is present. The first series of blankets may need changing every hour, or even oftener; the second, every 2 hours or so; and the remainder, every 6 to 12 hours. Much gold is lost through slovenliness in this department. (6) Sufficient care is not generally exercised in the selection of the fibrous material used for covering the strakes; and experiments with various kinds of hair and wool, and with different classes of fabric, may be expected to throw considerable light upon the subject.

From seventeen replies received to a circular recently addressed to Australian mine managers, it would seem that the inclination per

foot adopted for blanket tables varies from $\frac{3}{4}$ in. to $1\frac{1}{2}$ in., the average being about $1\frac{1}{16}$ in. White mentions that a fall of 1 in. per foot has been found to answer very well, and adds that blankets are useful in arresting free fine gold, and they may be used with good results where the material crushed is free from pyrites. Where, however, pyrites exist in quantity equal to only 1 per cent., it is advisable to employ concentrators; and where these are used, blanket tables become unnecessary, because the concentrators will arrest the free gold as well as the pyrites—thus discharging a double duty and rendering the blanket-table superfluous. Liddle remarks that the pitch of tables should be regulated according to the nature of the stone operated upon. Surface stone requires less incline than stone taken from below the water-level, or heavily mineralised stone, such as that usually found in the deep mines of Sandhurst.

Roasting.—The idea of roasting quartz before crushing was probably borrowed from the habits of aboriginal natives in many parts of the world, of burning rocks to facilitate their grinding in the imperfect appliances at hand. The plan first adopted by Australian miners was to roast the quartz in stacks in the open air, or in kilns, to oxidise the sulphides, and so liberate the gold, while the quartz was rendered more friable and easy to crush. After several years' trial, this system was given up, as it was found to be rather injurious than otherwise. At a low heat, the pyrites in the interior of the quartz was little changed, while the free gold was coated with a film of some material, probably sulphur, which impeded the action of the mercury on it. When the roasting was carried on with a higher degree of heat, the oxide of iron formed on the exposed faces of the quartz acted as a flux, and a glazed surface of slag was formed, in which numerous minute globules of gold could be discerned under the microscope, similar to those found in the waste tailings when crushing roasted quartz. In the interior of the quartz only a portion of the sulphur was given off, while black veins were formed by the melted mono-sulphide of iron; and other experiments led to the conclusion that a portion of the melted gold was diffused through these black veins in a form which rendered it more difficult to separate than when in its natural state.

But these objections do not apply to the roasting of quartz which is free from pyrites, and Wilkinson, as early as 1861, established a furnace, (p. 524) at Anderson's Creek, Victoria, for this purpose; this was the subject of a report, which may be summarised as follows.

The quartz subjected to this peculiar calcination is exceedingly friable, much more so than the best roasted quartz from open kilns. That whose treatment was witnessed by the Board was very crystalline and compact, and each piece of it after coming from the furnace could easily be broken by the hand, in every direction, into small fragments.

From the experiments made with a view of referring this important change in the hardness of the quartz to its legitimate cause, it seems to be mainly attributable to the high temperature (a moderate white heat) to which the quartz was subjected in the furnace. Wilkinson does not attach much importance to suddenly cooling, or quenching the heated quartz with water ; but the Board are of opinion that a difference does exist in favour of quartz thus treated over that which has been cooled slowly : this difference is greatest when the quartz is only heated to a dull-red, and becomes less marked when a bright-red or white heat is attained. It would also appear that the increased friability from this cause is due more to the explosive force of the steam than to the sudden change of temperature ; for it should be borne in mind that it is difficult, if not impossible, to effect the sudden cooling of masses of broken quartz on the large scale, even when the quantity of water available is very great, for the steam first generated forces the water from the interstices between the pieces of stone, preventing actual contact, so that the cooling is in reality much more gradual than it would at first sight appear to be.

The benefits to be derived from the friability of the quartz after calcination are very evident : much lighter stamps will suffice to reduce it than when the same is in a raw or imperfectly roasted condition. It is also obvious that grinding machinery of every kind will perform its office more efficiently and perfectly.

The high temperature obtained by Wilkinson gives rise to another phenomenon of equal or greater value, influencing more materially the ultimate result, viz., the change in the form of the auriferous particles. A close examination of the fine gold extracted after calcination has proved that it is all globular in form, or approximately so, and that the more minute the metallic particles are the more perfect is their spherical shape.* That this should be the case will not appear surprising, when it is remembered that the temperature to which the gold is heated is far above that necessary to effect its fusion, and that, being melted, the molecular attraction between its atoms must determine the spherical form, subject to the constraining influence exerted by the rigid material in which it is embedded. While the metallic particles diffused throughout the quartz are undergoing this change *in situ*, the heat also tends to diminish the cohesion which gives the quartz its hardness, to expand its mass, and to open the fissures in which the greater part of the gold is doubtless deposited, thereby aiding the desirable change, inasmuch as it lessens the constraining influences above referred to.

The Board are convinced that this change from the jagged or lami-

* A similar change in the form of gold, but to a very limited extent, affecting more particularly the larger particles, has been observed in gold from quartz roasted in the ordinary way.

nated form to the smooth and globular must exert a very beneficial influence upon subsequent washing and amalgamating operations, more particularly in preventing a loss of fine gold, for as a particle in the spheroidal form is contained within the least possible surface, it is plain that it will be less acted upon by moving water, or by mixtures of water and pulverised mineral matter, than when it is in a thin leafy or any other attenuated form. The gold will, therefore, sink more readily to the mercury, whereby a saving in time will be effected, and its amalgamation will be rendered more certain and perfect. At the same time, it should not be forgotten that the rounded particles will roll on hard and sloping surfaces, and modifications of some of the means in common use to prevent their escape may be found necessary.

The experiments made upon the gases used by Wilkinson demonstrate that when, as in his process, vapour of water is passed over red hot charcoal in excess, enclosed in a retort or large tube, two gases only are the ultimate result, namely, hydrogen and carbonic oxide: they have also shown that a dull-red heat is sufficient to effect the decomposition of the water, although the charcoal may be made much hotter with advantage. The manner in which the chemical changes take place is, probably, as follows:—The incandescent charcoal meeting with the steam effects the decomposition of the same by combining with its oxygen, forming carbonic acid gas, and setting its hydrogen free; the former of these gases is still further acted on by the large excess of charcoal present, it parts with half its oxygen to a fresh portion of carbon and gives rise to carbonic oxide; these two gases are remarkable for the heating powers they possess when subjected to combustion, more especially when, as in Wilkinson's furnace, a hot blast is used; they are in consequence well adapted for producing a very intense heat without the introduction of a large quantity of bulky fuel, and in a very short time. The following items have been calculated with a view of placing the circumstances under which these two gases are formed in a clearer light:—

One hundred parts of water by weight require 66·6 parts of charcoal to effect its complete decomposition and the production of the gases. The two gases are produced in equal volumes, weighing together 166·6 parts, while of this mixture 155·5 parts by weight are carbonic oxide and 11·1 parts hydrogen—or, in other words, 1 lb. avoirdupois of water (equivalent to about 28 cub. ft. of steam at 212° F.) will consume 10·65 oz. avoirdupois of charcoal, and generate 42 cub. ft. of the mixed gases, supposing the same to be cooled to the ordinary temperature of the air: so far, however, from being cooled, the gases enter the furnace at a very elevated temperature, and therefore much increased in bulk; and there meeting with the heated atmospheric air, all the conditions for rapid

combustion are established, while the heat is applied in a manner calculated to permeate the closely-packed quartz. There can be no doubt that a large portion of the heat in the furnace at Anderson's Creek was obtained from the solid fuel (wood and charcoal) directly introduced ; but the Board are of opinion that one half of this might have been saved by better management of the furnace and with more perfect mechanical arrangements for supplying the gases and blast.

It is probable that, besides acting as fuel, these gases play an important part as chemical agents in relation to the gold and other contents of the quartz, such as the metallic sulphides, arsenides, oxides, &c. Iron-pyrites, for instance, is reduced to metallic iron, strongly magnetic and containing but a small quantity of sulphur, an effect which could never be produced in an ordinary furnace charged with charcoal and supplied with a hot blast alone. In this case, there can be little doubt that the hydrogen is the active agent, and that its reducing power is greatest when, towards the close of the calcination, the admission of air is stopped, and the gases are still allowed to play upon the quartz. The metallic iron thus produced is, like the gold, found to assume the globular form ; the specimens of it which the Board had assayed were shown to contain gold in considerable quantities (in one case 0.03, and in another 0.07 per cent.), but whether more or less than the pyrites from which it was derived it is not possible to say.

Wilkinson claims for his process the credit of obviating the difficulty caused by the presence of sulphides with the gold : how far a further investigation of the peculiar reducing action of the hydrogen may tend to throw light upon this subject, the Board are not in a position to state ; but they do not think that Wilkinson can at present be said to have been practically successful in working out this particular point, for although in the case of iron-pyrites the sulphur has been almost completely expelled, still a certain proportion of gold is left in combination with the iron, and separate treatment will be necessary for its extraction. On theoretical grounds, the Board believe that an investigation of the effects produced by the mixed gases on arsenical pyrites, sulphide of lead (galena), and sulphide of antimony—minerals which interfere materially with the economical treatment of auriferous quartz—would lead to results of the greatest importance.

As far as the amalgamation is concerned, the Board are convinced that an imperfect roasting of the quartz, such as that commonly practised, is almost invariably detrimental. On the other hand, they are distinctly of opinion that the very high temperature obtained in Wilkinson's furnace would entirely prevent the coating of the gold with sulphur, arsenic, or other volatile matters, and that even if a deposit did exist, the action of the unburned hydrogen in the latter part of the process, would, in most

cases, tend to remove it, whereby the gold would be sent to the mercury with a pure metallic surface. In this respect, Wilkinson's process is a very great improvement on those in which the ordinary roasting is employed.

The question of cost, as far as the calcination is concerned, has been discussed by comparing that incurred at Anderson's Creek, with that of burning quartz in open kilns of 100 tons capacity at the Port Phillip and Colonial Gold Mining Co.'s works at Clunes.

The following table will show the result obtained.

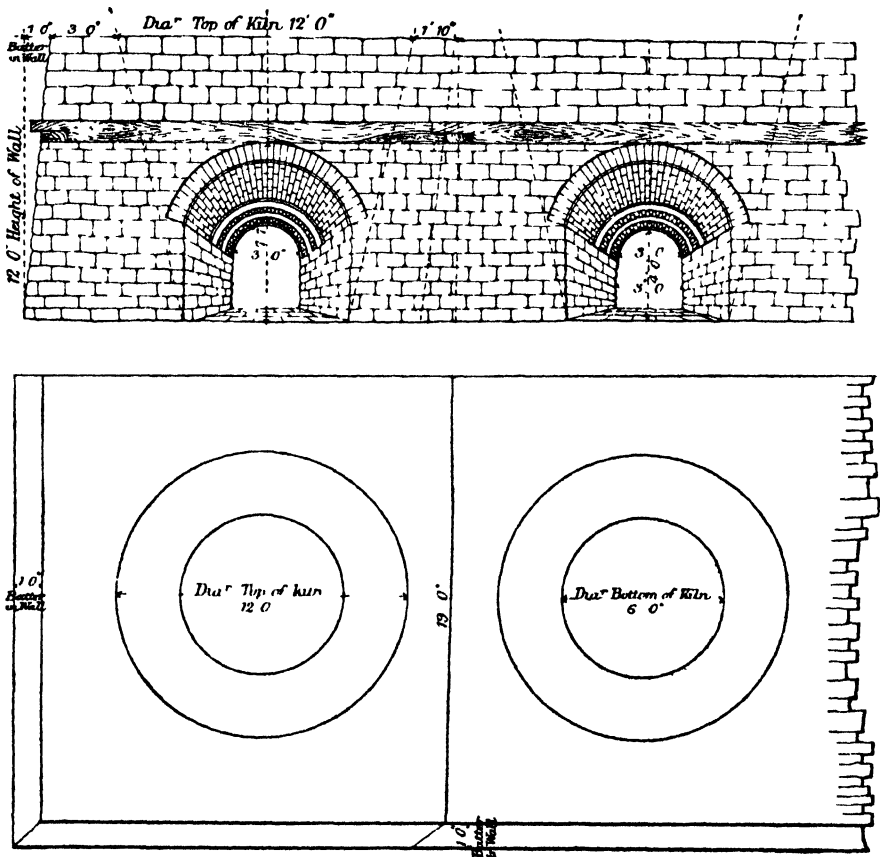
	Quantity of fuel per ton of quartz.		Cost of fuel.				Cost of labour per ton, charging and discharging.	Total cost.
	Wood.	Charcoal.	Wood.		Charcoal.			
	lb.	lb.	s.	d.	s.	d.	s.	d.
Anderson's Creek	455	84	0	9½	2	1½	2	8
Clunes	280	..	1	3	..	..	2	8
								5 7 3 11

In this table, the quantities of fuel are necessarily given by weight for both localities ; they must only be taken as approximate, as the weight of a ton measurement or cord of wood will vary in proportion as it is dry or wet, while charcoal becomes much heavier on exposure to the air. The heating power or efficiency of fuel will also depend on these circumstances ; the prices given are those which rule in each locality ; the cost of the gases used by Wilkinson has not been added, because it could not be estimated with any pretension to accuracy, and because the Board were convinced, that in the calcination they witnessed, it formed an inconsiderable item, the omission of which would not cause an important error. The cost of labour has been assumed to be the same in both cases. The Board have already stated their belief that half at least of the fuel introduced directly into the furnace might be saved by more perfect arrangements, and by calcining larger quantities at a time. They regard the solid fuel only as an accessory means of heating the quartz in the first instance, until the gases are properly ignited, and are convinced that for this purpose the charcoal might be altogether dispensed with, and that therefore the cost of Wilkinson's calcination might be considerably reduced.

The quantity of quartz operated upon was 1600 lb. weight ; it was very hard and crystalline, free from clay and "mullock," and was broken to about a 4-in. gauge. Only a speck or two of gold was discernible as it lay in the heap. The fuel used was 325 lb. of wood (peppermint and stringy-bark gum, tolerably dry) and 60 lb. of charcoal. Charging commenced in a cold furnace at 6 h. 30 m., a.m., and was completed at 7 h. 17 m. A good heat was then got up by the fan, and steam to the retort was turned on at 8 h. 15 m. ; at 8 h. 50 m. the gases were shut off,

b, furnace for hot blast and the gases ; *c*, generating and conducting pipes of hot blast ; *d*, retort to generate the hydrogen and carbonic oxide gases ; *e*, fire-bars ; *f*, conducting pipe of the gases ; *g*, chimney of gas-furnace ; *h*, roasting-furnace for quartz ; *i*, door for charging roasting-furnace ; *j*, platform for quartz and fuel ; *k*, truck for conveyance of calcined quartz ; *l*, steam-pipe conveying steam to gas-retort ; *m*, nozzles by which the gases enter the roasting-furnace ; *n*, nozzles by which the hot blast enters the roasting-furnace ; *o*, door of hot blast and gas furnace ; *r*, fire for production of hot blast and gases ; *s*, cast-iron bottom plate of roasting furnace ; *t*, fulcrum of lever ; *u*, tramway for conveyance of calcined quartz ; *w*, cast-iron support of gas-retort. A lever and windlass for opening and closing the bottom of the furnace are not shown.

FIG. 221.



QUARTZ KILN OF THE CALEDONIAN CO., MALDON.

In Fig. 221 is shown the arrangement of a kiln adopted by the Caledonian Quartz Mill, Maldon, for burning their quartz previous to

stamping. They have five of these kilns, each capable of holding about 50 tons. The burning of a charge occupies 9 days.

Some important trials have recently been made by R. D. Oswald, a mine owner at Maldon, for practically testing the value of roasting quartz before crushing, which point most impressively to its usefulness. In the trials the quartz was mixed in the mine before being brought to the surface, one truck having been taken for the roasted, and one for the raw process, alternately. In each case 180 tons were crushed. The yield from the roasted quartz was 202 oz. of smelted gold, or an average of 1 oz. 2 dwt. 10·66 gr. per ton. The raw quartz gave 169 oz. 11 dwt. equal to 18 dwt. 20·13 gr. per ton, the difference in favour of the roasting process being 3 dwt. 14·53 gr., or about 14s. 5d. per ton. From this must be deducted 6s. per ton, the estimated extra cost of the roasting process, the actual gain being, therefore, 8s. 5d. per ton. In addition to this, the wear and tear on the crushing plant is considerably less when the quartz is roasted; moreover, the stone is reduced in quantity by the process.

TREATMENT OF BLANKET-SAND.

Barrel Amalgamation.—The material gathered from the blanket-strakes, consisting of grains of free gold, globules of mercury, and particles of amalgam, which have been splashed or washed from the troughs, ripples, or plates, with a large quantity of pyrites and some worthless matter, is collectively termed “blanket-sand.” It is usually treated with mercury in a revolving barrel, the process being known as “barrel amalgamation.”

When the proportion of free gold is considerable, and the operation is properly conducted, it gives most satisfactory results. The barrels are of wood or iron, and are constructed to revolve on a pivot at each end. The charge is 8 to 10 cwt. of the damp blanket-sand, and 200 to 300 lb. of mercury. The charged barrel is set to revolve for about 8 hours, at a speed of 14 to 16 revolutions a minute. After this, it is filled up with hot water, and set to revolve again for another 4 hours, at a rate of 5 or 6 revolutions a minute. This concludes the operation, and the charge may then be drawn off. The free mercury and most of the amalgam are withdrawn first. The remainder of the contents is sent to a shaking-table, or some similar contrivance, for effecting the separation of the amalgam, pyrites, and refuse. The two latter are treated the same as the “tailings.” Stones and pieces of iron are sometimes put into the barrel, on the erroneous assumption that by grinding the sand finer they aid the amalgamation, whereas they are more likely to “sicken” the mercury by grinding the pyrites present. When the blanket-sand contains very little free gold, it is best treated like concentrated tailings, instead of by barrel amalgamation.

TREATMENT OF THE TAILINGS.

Definition and General Principles.—All the stamped material remaining beyond the portions which are caught by the blankets, plates, riffles, and troughs, is collectively known as “tailings.” It consists principally of fine earthy matters, but contains also more or less of gold, amalgam, mercury, and pyrites, chiefly the last. In some instances, especially when the ore has been largely pyritous, the tailings have a very high value; in all cases, the recovery of the gold held by them demands every attention from the miner who desires to achieve success. Unfortunately, their treatment is full of difficulty, and hence the only too frequent disposition to neglect them totally or partially. It is only where the poverty of the ores renders it imperative to extract all the precious metal they contain, in order to obtain a remunerative result, that any really effective and economical plans have been devised for the purpose. When the ores are sufficiently rich to pay a good profit from the more easily extracted portion, little attention has been given to this question. Yet there can be no excuse for throwing away a quantity of gold because the mine happens to pay well without it.

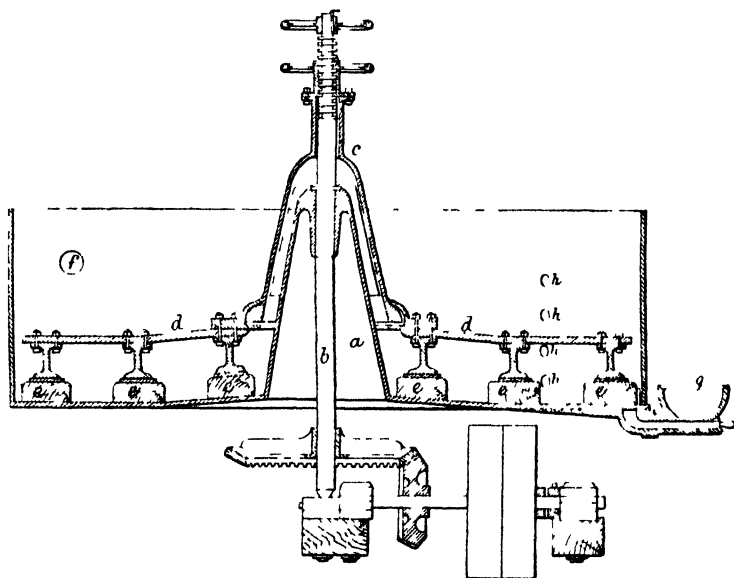
The treatment of the tailings should comprise three distinct operations, known as (1) “settling,” or allowing the heavy valuable matters to settle down in the mass; (2) “sizing,” or classifying the matters into several grades according to the size of the grains; and (3) “concentrating,” or still further separating the various materials according to their specific gravity.

The sands, pyrites, slime or slum, and water, which have escaped from the blankets and other devices and passed into the waste-trough, are sent into the “settlers,” where the heaviest portions fall to the bottom. The accumulated matters are cleared out periodically, and very frequently supplied directly to a concentrator. But this is a reprehensible practice, as is evident when it is borne in mind that all methods for the mechanical concentration of ores, based upon the fact that the ore is specifically heavier than the refuse, can only fully succeed when that superior weight of the ore is most thoroughly availed of. On a previous page allusion has been made to the circumstance that the relatively greater specific gravity of the particles may lose its effect in the presence of material of disproportionate size and shape. In other words, though pyrites is very much lighter than gold, and quartz is lighter still, yet it is easy to imagine pieces of pyrites or quartz whose size may be sufficiently great to counterbalance their relative lightness. The shape of the particles is determined by the crushing operation, and cannot be subjected to control; but the classification of the grains according to size,

at least within reasonable limits, is not difficult of attainment: indeed many contrivances are in use for the purpose in the concentration of such heavy ores as galena, tin-stone, &c. The value of these apparatus for treating auriferous pyrites is intensified by the fact that the gravity of the latter, as compared with that of quartz, presents far less contrast than does the gravity of lead- or tin-ore to its gangue; besides, the pyrites is so much softer and more brittle than the quartz, that it is crushed relatively much finer. For these reasons, the "sizing" of the tailings, before any attempt is made to concentrate their valuable portion, must evidently be of considerable advantage.

Settlers.—These differ somewhat in details of construction, but they usually are round tubs of iron, or of wood with cast-iron bottoms, resembling the pans in general features, but larger in diameter. A hollow pillar or cone *a*, Fig. 222, is cast in the centre of the bottom, within which is an upright shaft *b*. This shaft is caused to revolve by gearing below the pan. To its upper end is attached a yoke or driver *c*, that

FIG. 222.



SETTLER FOR TAILINGS.

gives revolving motion to arms *d*, extending from the centre to the circumference of the vessel. The arms carry a number of flows or stirrers, of various devices, usually terminating in blocks of hard wood *e*, that rest lightly on the bottom. No grinding is required in the operation, but a gentle stirring or agitation of the pulp is desired in order to

facilitate the settling of the amalgam and mercury. The stirring apparatus, or muller, makes about 15 revolutions a minute.

The settler is usually placed directly in front of the pan, and on a lower level, so that the pan is readily discharged into it. In some mills, two pans are discharged into one settler, the operation of settling occupying 4 hours, or the time required by the pan to grind and amalgamate another charge. In other mills, the settling is allowed only two hours, and the two pans connected with any one settler are discharged alternately.

The consistency of the pulp in the settler is considerably diluted by the water used in discharging the pan, and by a further supply, which in many mills is kept up during the settling operation. In other mills, however, the pulp is brought from the pan into the settler with the addition of as little water as possible, and allowed to settle for a time by the gentle agitation of the slowly revolving muller, after which cold water is added in a constant stream. The quantity of water used affecting the consistency of the pulp, and the speed of the stirring apparatus, are important matters in the operation of settling. Since the object of the process is to allow the mercury and amalgam to separate themselves from the pulp and settle to the bottom of the vessel, it is desirable that the consistency should be such that the lighter particles may be kept in suspension by a gentle movement, while the heavier particles fall to the bottom. If the pulp be too thick, the metal will remain suspended: if it be too thin, the sand will settle with it. Too rapid or too slow motion may produce similar results, because, by too violent motion, the mercury will not be allowed to come to rest on the bottom; while, if the motion be too slow, the coarser sand will not be kept in circulation.

A discharge-hole *f*, near the top of the settler, permits the water carrying the lighter portion of the pulp to run off; and, at successive intervals, the point of discharge is lowered by withdrawing the plugs from a series of similar holes *h*, in the side of the settler, one below the other, so that finally the entire mass is drawn off, leaving nothing in the settler but the mercury and amalgam. There are various devices for discharging these. Usually, there is a groove or canal in the bottom of the vessel leading to a bowl *g*, from which the fluid amalgam may be dipped or allowed to run out by withdrawing the plug from the outlet pipe.

The agitators through which the pulp passes after leaving the settlers are, in general, wooden tubs, that vary in size from 6 to 12 ft. in diameter and 2 to 6 ft. in depth. The main object in letting the stream of pulp pass through them is to retain and collect as much as possible of the mercury and amalgam and heavy particles of undecomposed ore that

are carried out with the pulp discharged from the settler. A simple stirring apparatus, somewhat resembling that of the settler, keeps the material in a state of gentle agitation, the revolving shaft carrying 4 arms, to which a number of staves are attached. In some mills, there are several agitators, in most cases only one, and by some they are not used at all. The stuff that accumulates on the bottom is shovelled out from time to time, usually at intervals of 3 or 4 days, and worked over in pans

Sizers. Sieves.—Probably the most exact contrivances for classifying according to size are inclined rotating sieve-drums. In these, the material is introduced at the higher end, and passes through finer and coarser sieves in succession. But the objection to their use is that they require a considerable amount of attention, so that they are not suited for a country where labour is dear; nor have they the capacity for treating the large bulk of material which accumulates at many gold-mines.

Fig. 223 shows an arrangement of six screens driven from a line of shafting at right angles by bevel gearing. The ore is fed into *a*; the finest separated matter passes through this screen, the remainder

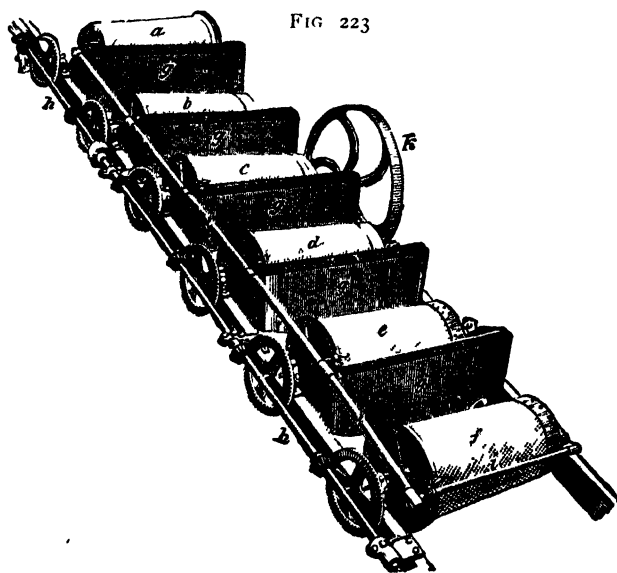


FIG 223

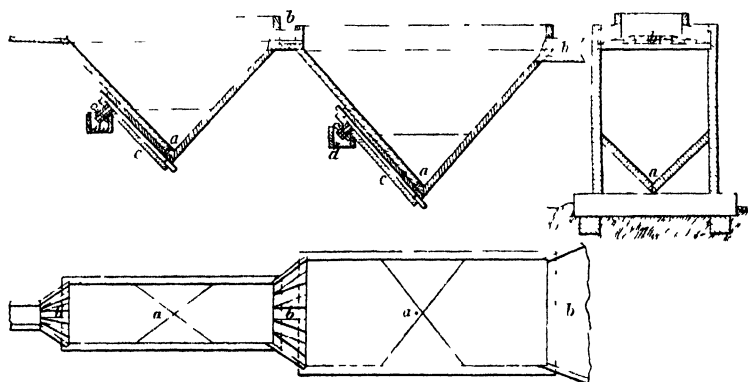
SERIES OF SIEVES

escaping at one end of *a* into the iron shoot *g*, which conducts it into *b* where the next finest sample is separated, and so on throughout the series. Such series of screens may be had from Calvert, Cornes, and Harris, 30, Walbrook, London.

Labyrinths.—The slime labyrinth is a German apparatus, and is used in some Hungarian gold-mills. It consists of a number of contiguous connected settling-pits, which, if, for instance, in connection with 3 batteries of 3 light stampers, increase in size from 1 to $1\frac{1}{2}$, $1\frac{1}{2}$, and $1\frac{3}{4}$ ft-square in transverse section, and respectively from 15 to 21, 24, and 36 ft. in length, with inclinations in the same sequence of $\frac{1}{2}$, $\frac{1}{2}$, and $\frac{1}{2}$ in. per foot the largest being horizontal. Such a labyrinth classifies the stuff into 4 portions, differing in size of grain, which form deposits during the passage from the smaller to the larger pits, the coarser grains depositing in the former. This classification is, however, far from perfect, and is besides attended with expenses on account of transport and re-puddling of the settled stuff, previous to further treatment. It also entails a large loss through escape and waste of fine material, that generally amounts to 10 or 12 per cent., but may, in unfavourable cases, rise to 15 or 20 per cent. The labyrinths are therefore now in operation only in some of the older and smaller establishments, where want of space or other circumstances prevent the use of either of the other two following classifying apparatus.

Pyramidal boxes.—The pyramidal boxes or *Spitzkästen* of the German gold miners, Fig. 224, are, as their name implies, hollow

FIG. 224.



GERMAN PYRAMIDAL BOXES OR SPITZKÄSTEN.

generally rectangular, pyramids. They are constructed of strong boards, well joined together (strong sheet-iron may be employed also). The sides are inclined at angles of not less than 50° , and there is a small hole in one side close to the apex. They are fixed horizontally, in an inverted position, and the crushed material is introduced at one of the narrow sides, a few inches below the top, by means of a launder. The result is that, as soon as the box is filled, a certain portion of the crushed matter

—i.e. the coarsest and heaviest, which the water, on account of its diminished velocity, is not able to carry farther—sinks and slides down the inclined sides of the pyramid, and escapes through the small hole *a* near the apex, whilst the finer and lighter matter passes off at the top by an outlet *b* in the centre of the side, opposite to the point of entrance. If now a second larger box be attached to the first, a third still larger to the second, and so on—each succeeding box at a slightly lower level, in order to prevent any settlement of stuff in the passage-ways—it follows not only that the same process of settling and escaping of the particles from the apex will take place in every box, but also that their size will decrease nearly in inverse proportion as the surface of a succeeding box is larger than that of the preceding one, or directly as the velocity of the water is diminished in it. According to this principle of the boxes—if they were made of only very gradually increasing size, and the apex holes proportionately small—it would be possible to classify the stuff into a great number of portions, different in size of grain, before it had entirely settled—i. e. till clear water passed off from the last box. Experience has, however, shown, that for fine ore-dressing in general, classification into 4 different sizes by an apparatus of 4 boxes is quite sufficient. The sizes of the different boxes, in order to ensure the most perfect classification, depend both on the amount of material which has to pass through them per second, and the size and character of the grains; and by theory and practice it has been found, that for the supply of every cub. ft. of material, the width of the first or smallest box must be $\frac{1}{10}$ ft.—i. e. for instance, for 20 cub. ft., 2 ft.—and for every succeeding box it ought to be about double that of the preceding one, or, generally, the widths of the boxes must increase nearly in geometrical progression, 2 : 4 : 8, &c., and their lengths in an arithmetical one, 3, 6, 9, &c.

For the stuff under notice, their dimensions are thus in different large establishments as follow :—

The first box is 6 ft. long and $1\frac{1}{2}$ to $1\frac{3}{4}$ ft. wide.			
„ second „	9	„	$2\frac{1}{2}$ to 3 „
„ third „	12	„	4 to 5 „
„ fourth „	15 to 16	„	8 to 10 „

Their depths depend on the angle of inclination of the sides, which, as already stated, is generally 50° , because if less, the stuff would be liable to settle firmly and choke the central orifice, and if larger, unnecessarily great height of the boxes would be required. The form of the two smaller boxes is commonly such that the two short sides are inclined at the above angle, and the two long ones, which would become far steeper, are broken—i. e. are for a certain depth from the top vertical, and afterwards inclined at the normal angle. This modification has, however, no

influence upon the action of the boxes, but simply facilitates somewhat their construction and firm fixing. The sides of the larger boxes are generally even throughout. The way in which the outlet-holes *a* at the apexes are constructed has an important bearing on the operation of the boxes. At these points, the hydrostatic pressure is considerable, and the holes should naturally be kept small, in order to prevent too much water passing with the particles of stuff; such small outlets are, however, especially in the treatment of coarser material, very liable to become choked. This difficulty has been met by the holes being made of conveniently large size, but connected with pipes *c*, $\frac{3}{4}$ in. in diameter, which rise up the side of the boxes—i. e. of the smallest box to within 3 or $3\frac{1}{2}$ ft., and of the others to within 2 to $2\frac{1}{2}$ ft. from the top—and are there furnished with small mouthpieces *d*, supplied with taps for regulating the outflow. This arrangement, on account of the outlets being so much higher, has the further advantage that a considerable amount of fall is gained (especially as regards the large boxes), which, for the subsequent treatment of the material, is in some cases of special value. There are two more points that require attention, in order to ensure good action of the apparatus, namely, the introduction of the material into the different boxes equally and without splashing, and prevention of the entrance of chips of wood, gravel, or other impurities that are likely to stop or obstruct the outlets. The first point is met either by having the supply-launders expanded fan-like and furnished with dividing-ledges *b*, or by the interposition of small troughs, the sides of which nearest the box to be supplied are perforated near the bottom by equidistant small holes. The cleaning of the material, previous to its entering the first box, is generally effected by the main supply-launder being made a little wider near the point of entrance, and the insertion at this place of a fine wire-sieve across the launder and somewhat inclined against the stream. This sieve must be occasionally looked after, to remove any impurities collected in front; and this, in fact, is the chief attention the whole apparatus requires, for otherwise it needs hardly any supervision. If once in proper working order, its action is constant and uniform, provided the material introduced does not change in amount and quality; and it has this further advantage, as compared with the slime labyrinths, that the classified stuff can, from the outlets, be directly conveyed in small launders to the concentration-machines for treatment, without any previous preparation. One point, however, not in favour of the apparatus, is that, having to be placed between the gold-mills and the concentration-machines, a great fall of ground is required, to permit the direct introduction of the material and allow sufficient fall for the tailings; and thus, where local circumstances are unfavourable, it has to be erected at a higher level, and necessitates the use of Jacob's ladders or pumps for

lifting the stuff. The action of the different boxes on the material under notice, with regard to the percentage of fluid matter and the quantity and character of its solid contents which they respectively separate, is according to experiment as follows :—

The small box separates 38 to 40 per cent., containing per cub. ft. 16 to 18 lb. coarse sand.

The second box separates 20 to 22 per cent., containing per cub. ft. 13 to 14 lb. fine sand.

The third box separates 18 to 20 per cent., containing per cub. ft. 15 to 16 lb. coarse slime.

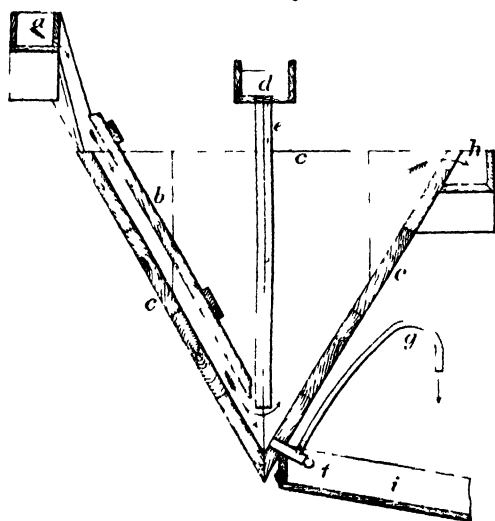
The largest box separates 10 to 12 per cent., containing per cub. ft. 10 to 12 lb. fine slime.

These results are satisfactory for the further concentration of the ore. As regards the loss caused through the final escape of impalpable ore from the last box, it amounts for rich galena-ores to about 6 per cent., and for quartziferous silver-ores to about $2\frac{1}{2}$ per cent.

According to J. M. Adams ('School of Mines Quarterly'), "several forms of pointed box are in use. Their dimensions vary according to the duty required. In some cases, it is desired to settle all the pulp, including the slimes, when there is too much water present for sub-

sequent concentration. In such event the pointed box should be about 6 ft. deep, and 3 ft. by 7 ft. at the top, the longest sides sloping till they meet at the bottom. Such a box will settle and save about 6 tons of ore in 24 hours, discharging it automatically and continuously from the bottom by a siphon hose, with the proper amount of water for subsequent concentration. This form is used when the tailings from pan amalgamation are to be concentrated, after leaving the settlers and agitators, for they contain a large excess of

FIG. 225.



POINTED BOX.

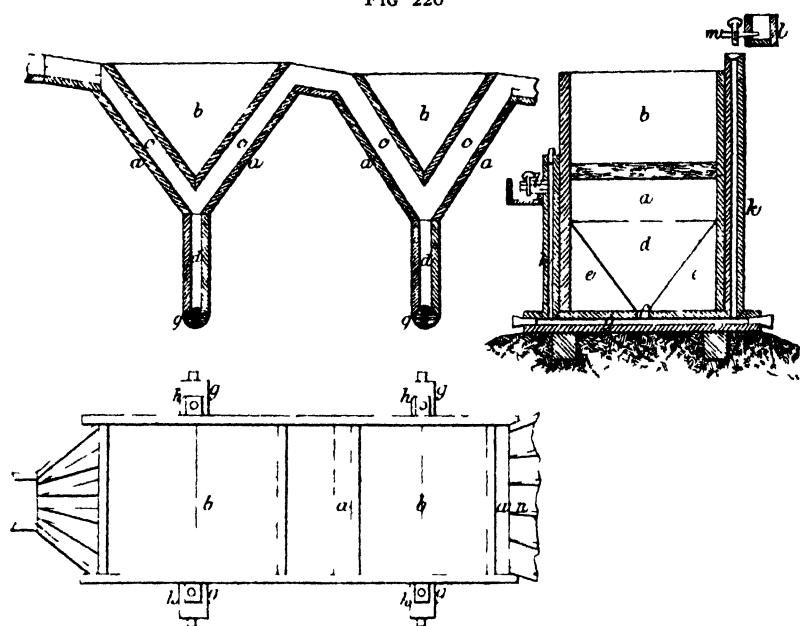
water, which must be got rid of, so that the tailings are of the proper consistency for concentration. Fig. 225 shows a form of pointed box used in cases where the slimes are to be separated from the battery pulp and saved. Each box is 40 in. square at the top, and 40 in.

deep, coming to a point at the bottom; and one box will handle from 6 to 10 tons of pulp in 24 hours, making a good separation. The pulp from the battery, entering the box *a* at the top, is confined by partition *b*, until it passes into the box proper *c*, near its bottom. Clear water is conveyed from a launder *d* above, through a $\frac{1}{2}$ -in. pipe *e*, which delivers it into the box at the bottom. Care must be taken that this pipe is kept full, so that no air bubbles are carried through it, as they create agitation, and cause sand, &c., to pass off with the slimes. The amount of clear water needed varies, so it is a good plan to have a cock in the pipe just below the clear water box *d*, or else to partially close, with a wooden plug, the opening of the pipe in the clear water box. At *f* is a hollow plug, and to it is attached a piece of hose *g*, which is used as a syphon, so that the pressure is lessened, and too violent discharge of the pulp is prevented. Without the syphon hose, $\frac{1}{8}$ -in. opening would not be too small, while with it $\frac{3}{8}$ -in. opening is about right, and the end of the hose is plugged accordingly. Inasmuch as foreign coarse material occasionally gets into the box (prevented as much as possible by a screen over the top), it is advisable to use in place of the hollow wooden plug shown, a $1\frac{1}{2}$ -in. iron T with one end plugged, and with $\frac{3}{4}$ -in. side outlet, attaching the syphon hose by nipple." The launder *h* carries off slimes, and the launder *i* conveys rich matters to the concentrators.

Triangular Double Troughs.—Classification in the triangular double troughs or *Spitzluten*, an invention by Rittinger, is based upon the principle that, if material composed of particles differing in size and density is exposed to a rising stream of water, the velocity of this stream may be so regulated that particles of certain size and character sink and may be conveyed off, whilst the remainder is carried upward by it; and that, consequently, by repeating this operation a certain number of times with a gradually decreased velocity of the rising stream each time, the material can thereby be separated in as many different classes of grains. The *Spitzluten* (Fig. 226), by which this action is now very simply produced, are constructed as follows:—Within a triangular trough *a*, of certain length and width, with two opposite sides vertical and two inclined at angles of 60° , is a similar smaller one *b*, having the vertical sides in common with the larger trough, but its inclined sides fixed at certain equal distances from, and parallel to, those of the latter. There is thus an open V-like space *c* left between the inclined sides of the two troughs, representing, as it were, a rectangular pipe, sharply bent in the centre; and it is through this that the stream of material has to pass—i. e. to fall and rise. The velocity of the stream depends on the size of this space, and consequently so does the size of the particles that will rise or sink in it. The cross-section and respective velocity stand in inverse relation

to each other, and their determination for each double trough of a complete apparatus is a matter of mathematical calculation, in which the size of the largest particles and the specific weight of the material to be classified form the main figures. For galena-ores, such as those under notice, and which are crushed so fine that the largest grains are not more than 0.6 millimetre in diameter, the most satisfactory classification into 4 different kinds of grains is, according to Rittinger's calculation, arrived at by a series of four double troughs, with the velocity of the stream decreasing from the first to the succeeding troughs in the progression of 2.3, 0.94, 0.37, 0.15 in. per second, and if the width of the channel for

FIG 226



TRIANGULAR DOUBLE TROUGH OR SLUICING.

the first trough is 1.1 in., and its length 2 ft., the dimensions of that of the second trough follow as 2.75 in. : 2 ft. And as it is not advisable to increase the width of the channels beyond 3 in., the channels of the third and fourth troughs are each 3 in. wide, and respectively about 54.5 in. and 135 in. long. The mean depth of the channels, measured from the line of inflow of the material to the lowest part of the inside trough, is, for the two smaller double troughs about 3 ft.; for the two larger ones, 4 to 6 ft. In order to carry off the coarse particles that sink in the channels, the inclined sides of the outside troughs do not meet below, but are continued downward, forming a long and narrow pyramidal opening *d*, about 1½ in. wide at top. The short sides *e* slope inwards at an angle

of not less than 50° , contracting the opening to a small hole f of about 1 in. sq. at bottom, through which the material is discharged into a horizontal pipe g , that extends both ways a small distance beyond the sides of the apparatus, and is connected at the ends with vertical 1-in. pipes. One of these, h , serves for the outlet of the classified material, and is carried up to within 36 to 21 in. of the water-level in the channel c , according to the degree of fineness of the particles that have to pass through it (the same as in the pyramidal boxes). At the top it is supplied with a tap for the regulation of the outflow. The other pipe k conveys a supply of clear water, furnished from a launder l supplied with a tap m , and as the water in the pipe stands 6 to 8 in. above the water-level in the trough, a small uniform pressure is produced, causing a forced influx of water at the point f , which is essential for good classification. This water—opposing itself to the downward current, charged with sediment in the pyramidal channel d —prevents all but the coarser particles and pure water passing into the pipe h , and thus only grains of the desired size are carried to the outlet i . With regard to the relative positions of the different double troughs of the series, they are fixed exactly horizontal, and sufficiently below each other to prevent any settlement of material in the communication-launder n , which are necessarily very broad. Other particulars regarding proper working, supervision, &c., are the same as those given for the pyramidal boxes. According to present experience, a series of 4 of these double troughs classifies as well as, and, for the two coarser kinds of grains, even better and cleaner than, a set of 4 pyramidal boxes, though for the fine slimes these latter are generally preferred, as they effect the desired settlement of the stuff more completely. A complete apparatus of troughs requires also less fall and space than one of pyramidal boxes, and is more easily regulated in cases of increased or diminished influx of material. The necessary additional supply of clear water might, however, form a drawback to its application in cases where this medium is scarce. As regards the results of classification by the different troughs of the series, they are stated to be as follows:—The first or smallest trough separates about 30 per cent. of coarse sand; the second, about 25 per cent. of fine sand; the third, 20 per cent. of coarse slime; the fourth, 15 per cent. of fine slime.

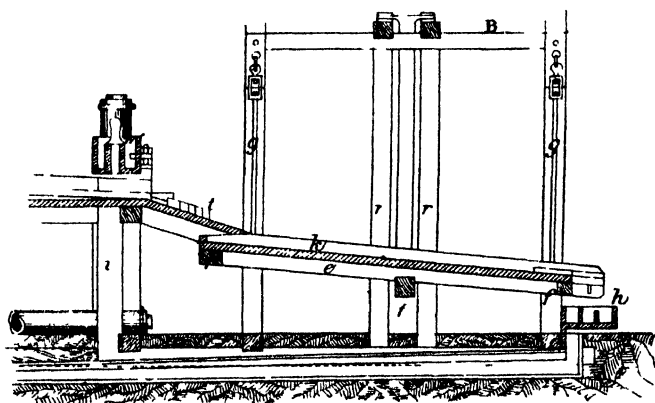
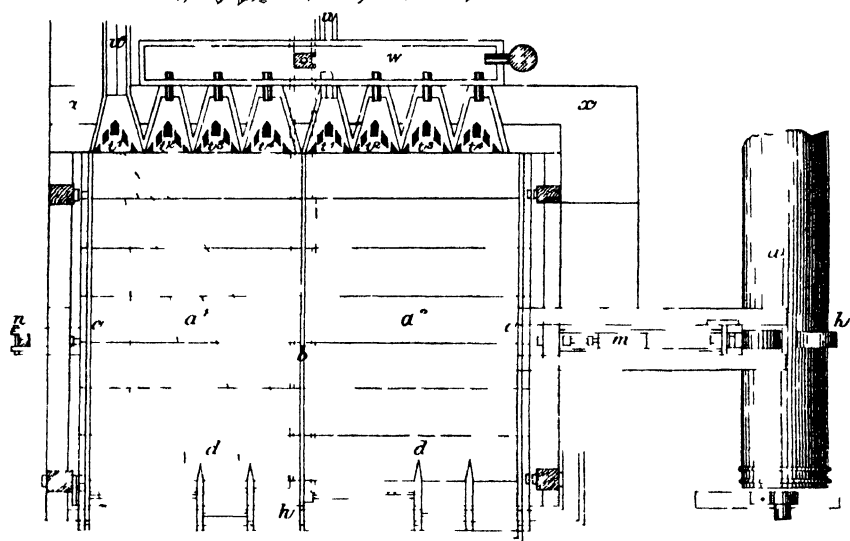
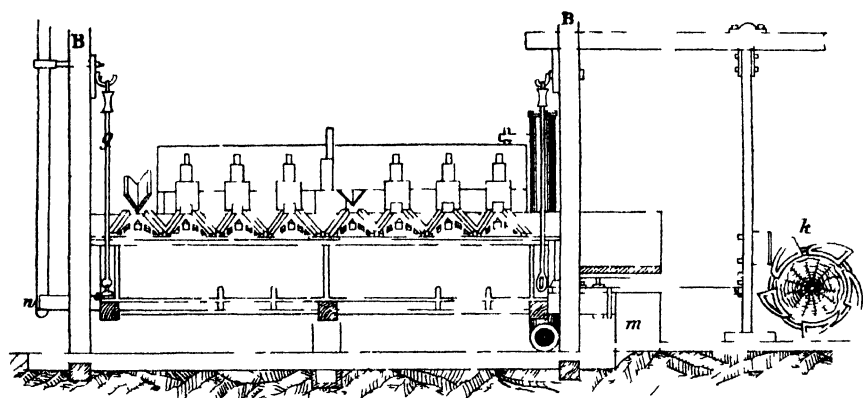
Concentration.—Having classified the material according to size, the next step is to submit each separate size to a process of concentration with the object of eliminating the valuable portion. For this purpose many apparatus are in use, all working upon the principle of taking advantage of the greater specific gravity of the part sought to be saved.

Percussion-tables.—The most highly perfected of the various per-

cussion-tables or shaking-tables is Rittinger's continuously-acting side-throw percussion-table, shown in Fig. 227.

To simplify the construction and movement of these tables, they are generally made so that they represent one large table, divided by a cheek *b* in the centre into two (a^1 and a^2), for the movement of which consequently only one arrangement is required, rendering the percussion simultaneous for both. The floor or platform of each table (a^1 and a^2), measured inside the head-board and cheeks *c*, which are about 4 in. high and $1\frac{1}{2}$ in. thick, is 8 ft. long and 50 in. wide. It is generally double boarded, the upper surface being made of tongued-and-grooved $1\frac{1}{2}$ -in. boards of some even, close-grained wood (generally sycamore), planed as exactly as possible, and slightly blackened by weak sulphuric acid. The boards are carefully laid crossways, and fixed with wooden pegs to the lower floor, made of pine-boards tightly screwed to a stout wooden frame, consisting of 4 or 5 bars *e* lengthways, and 3 *f* across, which are mortised and screwed together and secured by iron angle-braces. The centre crossbar *f* is nearly double as strong as the others, and projects on both sides a certain distance beyond the platform. It is called the "tongue" or "percussion-bar," as it forms the part to which the side-movement and percussion of the table is imparted. The double table is suspended by 4 iron rods *g*, having adjusted shackles, and at either end eyes that are connected with hooks; the upper ones screwed into stout uprights, that form part of a strong framing *h*, braced well together at top and bottom; the lower ones screwed into the sides of the platform frame at about 1 ft. from either end. The arrangement for imparting the side way motion and percussion to the tables consists, in the first instance, of a wooden axle *a*, furnished with 4 or 5 cast-iron cams *k*, opposite the centre of the table. This axle is turned by an endless strap from a shaft connected with the axle of a water-wheel. The cams act upon—i. e. push forward—the iron-faced projection of a vertically-suspended wooden lever, which swings at its upper end on small iron pivots between two crossbars connected with the framing whilst its lower end moves between guiding ledges, nailed to the floor of the building. About level with the frame of the table the ends of 2 wrought-iron bridles are joined to it by means of a screw-bolt. These bridles transfer the forward movement of the lever to the table by being fixed with a screw-collar over a horizontal screw-spindle, that is fastened and adjusted on top of the projecting portion of the percussion-bar or tongue, and by means of which the length of the forward movement—i. e. the side-throw—can be regulated. The end of the projection of the percussion-bar, being slightly rounded off in front and strengthened by an iron rim, presses, when at rest, against the bumping-block *m*—a stout square pillar—which is joined to a foot-piece, 5 or 6 ft. deep in the ground, and

FIG 227



RITTLER'S PERCUSSION-TABLE.

well stayed at the back to resist the shocks, and its face, at the place of contact with the percussion-bar is generally covered with leather for the purpose of deadening the blows. The pressure of the percussion-bar against the bumping-block is produced by a stout spring *n* of iron or some tough wood, attached to its prolonged end at the opposite side of the table, and can be regulated by a screw, that adjusts the tension of the spring. Both prolongations of the percussion-bar move on each side between two uprights *r*, connected with the outside framing, by which means the transversal movement of the double table is guided.

The following is a somewhat more simple arrangement for moving the table, and is similar to that used for the common percussion-table. The cams of the driver are made to act against the iron-faced top end of a lever, that moves on an axle at foot, and to which is attached, by means of a regulating screw, a horizontal wooden bar. This slides between guides, fixed on top of the bumping-block, and at the other end is in contact with an oblong block of hard wood, screwed on top of the prolongation of the percussion-bar. In order to prevent the horizontal bar and lever from jumping back too far when the table strikes against the bumping-block, the bar is furnished with a bolt, which ensures the normal position by resting against guiding-ledges on top of the bumping-block. The other arrangements, as regards the spring, regulation of pressure, &c., are the same.

The tables receive their supplies of classified material and of a necessary amount of cleaning water, evenly distributed by means of triangular inclined dividing-planes *t*¹, &c.,—furnished with wooden buttons in the usual manner,—from separate troughs *w*, into which the material and water are conveyed by small launders or pipes *u*, from the respective classifier and main water-launders. The whole of this arrangement, inclusive of platforms *x* for the workmen to stand on, rests on a low framing *i* above the head of the table. There are generally 3 or 4 dividing-planes *t*¹ to *t*⁴ for each table, one of which supplies the classified material for a breadth of 8 to 10 in. at that side of each table opposite to where the percussion takes place, whilst 2 or 3 others provide the cleaning or washing water, the one nearest the percussion side always in somewhat larger quantity.

The principle upon which the ore-concentration on these tables is based, differs from that of the common percussion-tables mainly in the side action of the percussion on the stuff treated, which produces two movements of the particles, viz. one down the incline, the other forward; and in the mean direction resulting from this—i. e. diagonally downward,—they pass off the table. As now the heavier particles—i. e. the ore—are not only thrown farther, but are also, on account of their stronger

friction on the boards, more or longer exposed to the forward movement than lighter ones (waste) during the same interval of time, it follows that they travel outside these, gradually separate according to their specific weights into distinct bands, and that this separation becomes more perfect, the nearer they approach the end of the table. The whole of this action is at the same time enhanced by the "cleaning-water." This prevents the stuff from spreading at once all over the table; it cleans the outside streaks of ore, and the stronger portion serves specially for washing the ore finally off the apparatus. For securing the partitions of the different portions of ore, separated on the table, small, pointed, movable pieces of wood *d*, called "tongues" or "pointers," are screwed upon the table near its foot, by means of which the streaks of ore are divided from that of the waste, and guided through narrow slits in the table, furnished with sheet-iron lips, into separate launders *h*, underneath, that discharge into settling- or catch-pits; or else the ore passes off the table over small movable strakes into launders placed in front. The waste runs off the table in both cases over a broad strake into a small launder in front, that conveys it to the main waste channel. In this, and in all instances where the discharge takes place over the front of the table, it is, like the slits above mentioned, provided with sheet-iron lips, projecting about 2 in., in order to prevent the stuff licking back underneath.

The result produced by the operation of these tables on any of the four previously classified portions of the stuff under notice is, to divide them into five products, viz:—

1. Lead-ore, containing fine free gold.
2. Lead-ore, poor in gold.
3. Copper- and iron-pyrites, mixed with some lead-ore.
4. Poor ore-slime (i. e. imperfectly concentrated); and,
5. Waste.

These run in as many well-defined streaks down the tables, and are, as above described, parted by the pointers *d* and guided, the first three, into launders communicating with separate settling-pits; the fourth, into a strake, that conveys it to a catch-pit, from whence a Jacob's ladder, or lifting-wheel, raises it into a small pyramidal box for re-classification—the portion issuing from the apex orifice being then conveyed to, and re-treated on, a separate table. The quality of the three first portions of ore collected in the settling-pits is such as to render them fit for direct metallurgical treatment, the auriferous portion being, however, previously submitted to gold extraction, as will be seen hereafter. There are many conditions, as regards adjustment of stroke, supply of material and washing-water, &c., necessary to ensure the satisfactory working of this machine. This is shown by the following

table (prepared by Franz Rauen), which gives the results of practical experience in treating the four classified portions of sands and slimes, both of auriferous lead-ores and silver ores.

ADJUSTMENTS OF RITTINGER'S CONTINUOUSLY-ACTING SIDE-THROW PERCUSSION-TABLE.

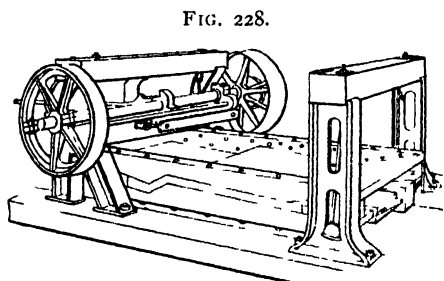
Description of Ores.	Inclination of the Table.		Constant pressure of table against bumping-block.	Throw or stroke.		Supply of material and cleaning water per minute upon a double table.						
	In the direction of the throw.	Towards the front.		Length.	Number of throws per minute.	Classified Material.			Cleaning Water.			
						Supply in measurement.	Weight.	Solid contents carried by the water.	Front cleaning water.	Back cleaning water.		
FOR AURIFEROUS LEAD-ORES.			Lines.	Line.	lb.	Lines.	No.	Cub.ft.	lb.	lb.	Cub. ft.	Cub. ft.
Coarse Sand, 1st Classifier.. ..	..	84	140	24	73	0'392	23'51	2'665	1'023	1'251		
Fine do. 2nd do.	6	58	110	21	85	0'333	20'09	1'763	1'093	1'234		
Coarse Slime, 3rd do.	7	56	106	18	100-105	0'323	18'94	0'914	0'888	0'977		
Fine do. 4th do.	14	52	100	10	112-130	0'236	14'07	0'750	0'669	0'914		
FOR SILVER-ORES.												
Coarse Sand, 1st Classifier.. ..	6	72	212	18	76-78	0'420	26'64	4'956	0'771	1'187		
Fine do. 2nd do.	6	54	181	12	86-88	0'405	24'80	3'410	0'256	0'677		
Coarse and Fine Slimes } 3rd and 4th Classifiers	6	30	100	10	100-110	0'261	16'55	3'580	0'373	0'708		

Another condition, not less important than those just given, for good concentration of the different classified portions of material, is the proper regulation of the velocity of the throw. For the tables applied to the treatment of the two coarser sizes, it ought, in the average, not to exceed 1 ft. per second, whilst slime-tables require a velocity of stroke of only 0'5 ft. per second; a greater velocity causes the tables to slide, so to speak, from underneath the particles of ore, thus retarding their progress.

The motive power required for working a double table is, by dynamometric experiments proved to be, 0'26 H.P.; and the working effect per hour of a single table is 55 lb. of slimes and 300 lb. of sands. Six continuously-acting double tables are capable of working in 24 hours 10 to 12½ tons of crushed material, classified by four pyramidal boxes. Four of these treat the four classified portions—i. e. each table is devoted to one particular class—and the two remaining ones are necessary for reworking the intermediate products—i. e. the imperfectly concentrated ore matter. As regards supervision and manual labour required, one workman is quite sufficient for attending on two properly-adjusted double tables. His principal work consists in looking after the right position of the "pointers," and the steady and regular supply of material and cleaning-water. On comparing the relative quantities of ore produced and the loss sustained during the same time by this and

the old percussion table, the produce of the former is 3 to 4 per cent. smaller than that of the latter, and its loss of ore about 2 per cent. larger—i. e. the new table loses 23 to 24 per cent., whilst the old one only loses 21 to 22 per cent. These disadvantages of the new invention are however, more than compensated by the greater purity of its ore effecting a saving both in transport and smelting expenses, and more especially by the greater amount of free gold contained in the auriferous portion. But it greatly excels the old table on account of its continuous, steady self-action; and consequently its working expenses are, without regard to wear and tear, fully 60 per cent. lower than those of the other.

Ordinary percussion table.—The end shake percussion table shown in Fig. 228 is a simple and inexpensive form of concentrator largely used in the Australian reduction works, and, when carefully handled, gives exceedingly good results with ores which do not offer special difficulties in concentration. It will be seen from the engraving that the wooden box into which the pulps flow is suspended by iron rods from cross-beams carried on the top of the upright standards, and, by the action of a cam driven by the pulley on the cam shaft and working against a cast-iron block



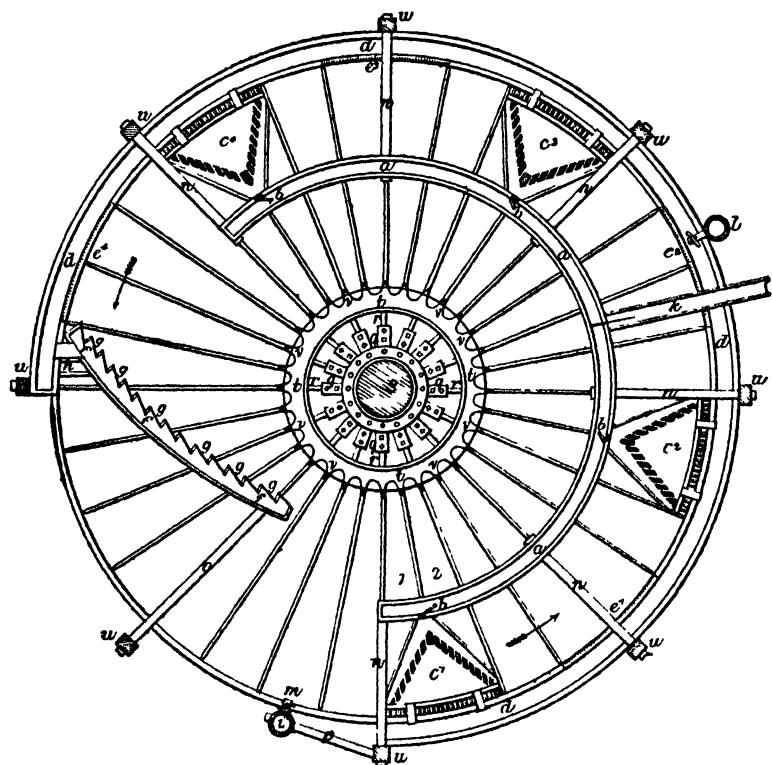
AUSTRALIAN PERCUSSION TABLE.

which is fixed on the lower end of the box or table, a succession of sharp blows is given, causing the sulphides and heavier portions of the crushed ore to settle to the bottom, while the lighter material is carried away by the flow of the water. Where close concentration is necessary, special machines are undoubtedly better in every way; but this old-fashioned shaking-table is still preferred in some districts on account of its simplicity and the ease with which it can be repaired. It may be had of Charles Appleby & Co., 89, Cannon Street, London.

Rotating table.—Rittinger's rotating table, Fig. 229, is specially applicable for the concentration of fine slimes, and for this operation is preferred to both the common and side-thrown percussion-tables. The concentrating portion—i. e. the table proper—may be described as a shallow, inverted-conical or flat funnel-shaped ring, consisting of even-grained, well-planed 1-in. pine-boards. The outer diameter is 16 to 18 ft., and the inner 5 to 6 ft., with an inclination of 6 in. to its radial width. It is furnished round the outer periphery with a rim of board 2 to 3 in. high, and is divided radially by narrow battens into 32 equal

segments, that are somewhat contracted at the inner periphery by the ends of the battens being split, where, attached to the end of each segment, is a funnel-shaped descending-pipe of wood or sheet-iron *v*, serving for discharge into receptacles beneath. This ring rests exactly horizontal, or rather the boards which it consists of are fastened horizontally crossways and watertight, upon 16 radial wooden bars or arms *r*, attached by means of a cast-iron rosette *q* to a central vertical wooden axle *s* of 16 in. diameter, to which (and consequently to the

FIG. 229.



RITTER'S ROTATING TABLE.

table) a slow, steady revolving motion is imparted by means of a tangent-screw, from a shaft, connected with the water-wheel. The tangent-screw operates upon a finely toothed cast-iron wheel, about $2\frac{1}{2}$ to 3 ft. in diameter, fixed on top of the axle. The lower pivot of the latter turns in strong cast-iron bearings, whilst the upper one revolves within an iron collar; the bearing and collar, being fixed to a strong framing, consisting of 8 radial spars at top and bottom, connected, about 1 ft. outside the circumference of the table, by as many stout uprights *u*, and strengthened

above and below by 4 braces, forming squares, near the centre. To this framing are attached all supplementary portions of the table, that take no part in its motion, such as the troughs for supply of clear water and material, triangular distributing-planes, &c.

The circular trough *a* that supplies the material consists of boards or sheet-iron, is 3 to 4 in. wide, and extends over 20 segments of the table. It rests horizontally on supports *n*, fastened to the uprights *u* of the framing, within about $2\frac{1}{2}$ ft. of the outer periphery of the table, and at such a height above it that the triangular distributing-tables *c*, that introduce the material close within its rim, have a fall of at least 20° . The bottom of the circular trough, from the point where a launder *k* introduces the material from the classifier—i. e. just at the centre of the curve,—is saddle-backed at a fall of about $\frac{1}{2}$ in. per ft., which prevents the stuff from settling. Up the centre of the mouth of the launder *k* a small movable tongue of wood is fixed to distribute the material equally towards both sides of the saddle. There are generally 4—sometimes 5—square openings *b* through the outer rim at the bottom of the trough, also provided with tongues, to regulate the outflow of the material on to as many triangular distributing-tables *c*^{1, 2, 3, 4}, the relative positions of the openings being such that one is near each end of the trough, and the other 2 or 3, as the case may be, are placed respectively at equal distances from these and between each other. As regards the triangular distributing-planes *c*, they are fastened with their pointed ends by means of bolts and eyes underneath the trough, whilst their lower edges have hook-like strips of sheet-iron attached, by means of which they hang firmly on the inner rim of the main clear-water trough *d*, presently to be described. This arrangement, though ensuring a fixed position of the planes, yet permits their easy removal in cases of access to the table being required. They are, as usual, constructed of pine-boards, to which oblong, wooden, distributing-buttons are fixed; but they have, for the more equal distribution of the material, attached to their lower edges either pieces of sheet-iron with serrated lips, or are provided harrow-like with a number of thin iron spikes. The width of these lower edges which are concentric with the table, is $\frac{1}{16}$ of the circumference of the latter—i. e. each supplies 2 of the 32 segments at a time.

The main clear-water trough *d*, constructed of boards or sheet-iron, is likewise circularly bent, and rests horizontally, 3 or 4 in. above the circumference of the table, on flat iron supports projecting from the uprights *u* of the framing. Its inner rim lies just within the rim of the table; the outer one touches the uprights, and is attached to them. It commences radially abreast of one of the ends of the feeding-trough *a*, just level with the first distributing-plane *c*¹, but it extends 4 segments beyond the last distributing-plane *c*⁴ at the opposite end of the trough *a*

and encompasses in all 24 segments of the table. It is constantly and in regulatable quantity kept supplied with clear water by means of a tap *b*. The amount required for concentration runs from the trough on to the circumference of the table through open cuts e^1, e^2, e^3, e^4 in its inner rim, $2\frac{1}{2}$ in. deep, and as wide as suffices for the supply of 2 segments of the table. For the equal distribution of the water, sheet-iron plates, deeply serrated on both edges, are fixed in front of these cuts (indicated by dark lines on the sketch.) The teeth of the lower edges of the plates stand about $1\frac{1}{2}$ to 2 in. off the surface of the table, whilst those of the upper edges reach to within $1\frac{1}{2}$ in. of the level of the outer rim of the trough. The number of the places of overflow $e^1, 2, 3, 4$ corresponds with that of the distributing-tables *c* furnishing the material, and they are so arranged that if, for instance, there are 4 such tables, 3 overflows e^1, e^2, e^3 occupy the centres of the spaces—i. e. they supply the 2 middle segments of the 4 between these tables, whilst the fourth e^4 lies 2 segments beyond the fourth table. Adjoining this last place of outflow, and communicating by a short spout *h* with the main trough *d*, a peculiarly-constructed special clear-water trough *f* commences and extends horizontally in a flat curve across 6 segments of the table to near the inner periphery of the latter, where its end rests on a strong support *o*, projecting from the nearest upright *u* of the framing. The outer rim of this trough is even, but the inner one presents a succession of deep notches or breaks, and the dark lines *g* signify the places lower than the rest, where the clear water flows over the serrated sheet-iron plates, that extend down to within 2 in. of the surface of the table. And as they have to follow the inclination of the table, whilst the trough lies level, they become gradually longer towards the end of the trough. The special purpose which this trough serves for the material under notice is the separation of the pyrites from the lead-ore. As regards the arrangement for washing this ore off the table, it consists of a vertical pipe *i*, that by means of a flat mouthpiece *m*, 10 lines wide and 3 lines high, discharges, under a pressure of 8 to 9 ft. and at a very oblique angle, a stream of clear water on to the table near its rim. This pipe is closed at the bottom, and communicates at the top with a reservoir or launder. It rests on a support *p* projecting from the nearest frame-upright *u*, close outside the circumference of the table, and in the centre of the space between the commencement of the main clear-water trough *d* and the end of the special one *f* just described, a spot coinciding with the centre of the middle one of the 3 remaining segments of the table.

For the separate reception of the three kinds of material—viz. waste, pyrites, and lead-ore,—that run off the table at its inner periphery (through the funnel-shaped pipes *v* attached to the contracted ends of

the segments), either one circular trough *t* divided by 2 partitions into 3 compartments, or else 2 separated troughs, one within the other, are used, the outer one of which is devoted to the waste, whilst the inner one receives the 2 kinds of ore into separate compartments by means of 2 short, shallow strakes, resting on top of the outer trough. The positions and lengths of the respective compartments correspond, of course, with the number and positions of the segments of the table, from which waste, pyrites, and lead-ore are washed off. The troughs rest on the foundation-bars of the framing, and the compartments for the 3 products communicate by holes in their bottoms with separate launders underneath, that conduct the ore to settling-pits outside the table, and the waste into the main waste-channel.

The mode of action of this table is very simple, and its working requires no manual labour whatever; the machine is, in fact, like the percussion-table—self-acting. The process of concentration on the two segments 1 2 is, for instance, as follows:—Being coated with material by the triangular plane *c*¹, they pass during the slow rotation of the table underneath the first overflow *e* from the main clear-water trough *d*, and the ore deposited on them is cleaned from the waste; but, progressing farther, they receive a fresh supply of material from the second distributing-table *c*², are again cleaned by the second overflow *e*² of clear water, and after having undergone this double operation twice more, they progress underneath the special cleaning-trough *f*. On account of the inwardly-bent form of this trough, the numerous streams of clear water, issuing from it at *g*, act gradually on nearly the whole surface of the segments, completely washing off the pyrites and leaving them coated with pure lead-ore only, which, on farther progression, is also removed by the forcible stream of water, discharged from the mouthpiece *m* of the vertical pipe *i*. With this operation a full rotation of the table is completed, and the whole process, as regards the 2 segments, commences again. But it will no doubt be understood, that every 2 segments composing the table undergo the same operation in steady regular succession, and that the concentration is thus continuous as long as the table rotates.

The conditions necessary for satisfactory working of the machine are the following:—

1. The supply of material for 4 to 5 simultaneous coatings ought not to exceed 0·6 to 0·7 cub. ft. per minute.
2. The solid contents per cub. ft. of supply must not amount to more than 1 to 1·25 lb.
3. 2·25 to 2·40 cub. ft. of clear water are required per minute, i. e. 0·75 to 0·80 cub. ft. for separating the lead-ore from waste and pyrites and 1·5 to 1·6 cub. ft. per minute for washing it off the table.

4. The table must not rotate faster or slower than once every 10 minutes, i. e. 6 times per hour.

Comparative experiments touching the relative merits of this machine and the old percussion-table for the treatment of slimes, have proved that, whilst on the latter an amount of material holding 45 to 50 lb. of solid contents can per hour be passed through all stages of concentration to that of pure ore; the rotating-table accomplishes the same result in like time from a supply of similar stuff containing 90 to 120 lb. of solid matter—i. e. it concentrates fully double the quantity in the same time. The concentrated ore from both machines is equal, as regards purity, &c.; but comparing the relative amounts produced, viz. 80 per cent. by the new and 72 per cent. by the old one from an equal quantity of material, it follows that the old table loses 8 per cent. more than the new one.

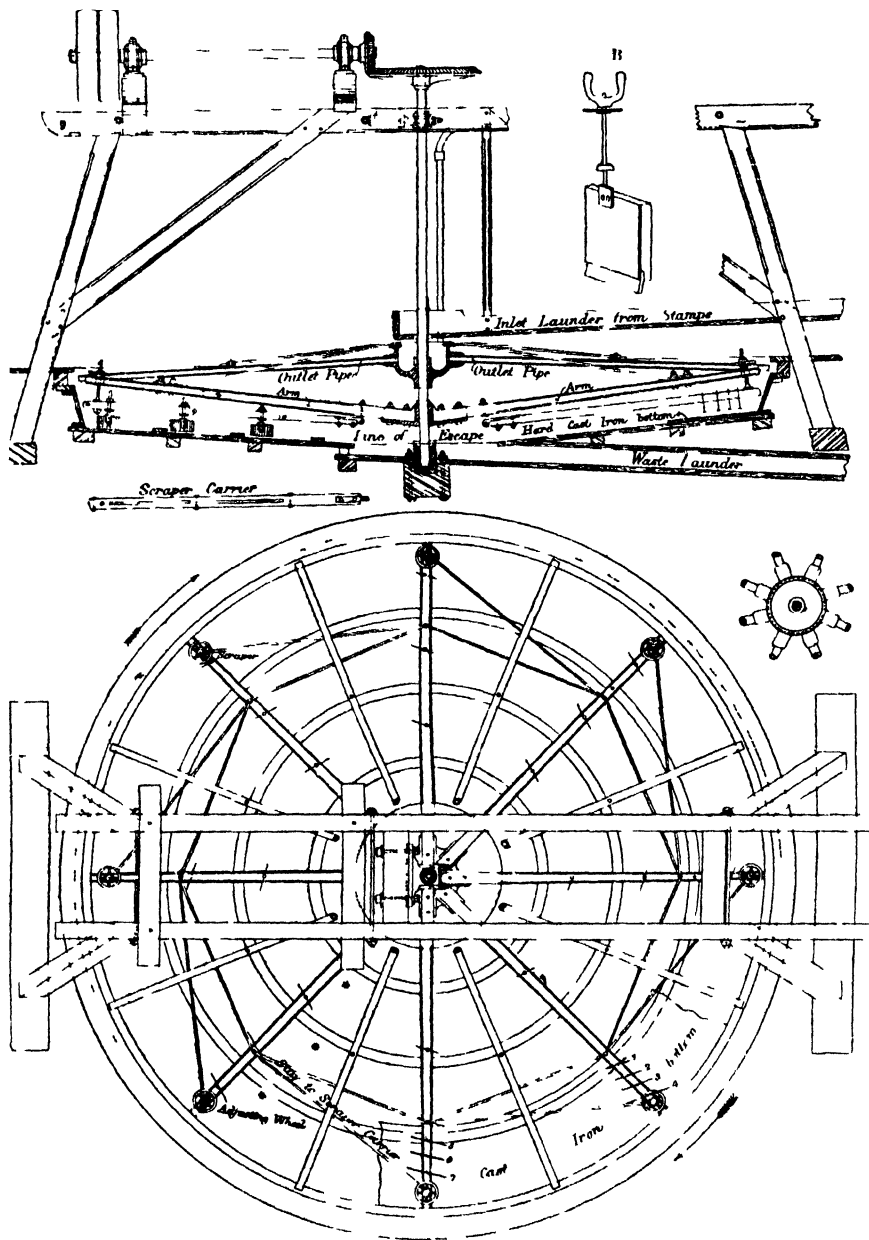
As regards working expenses, the latter has also an important advantage over the former; for its supervision can be accomplished by a boy, whilst the manipulation on the percussion-table requires a strong and experienced person, whose wages are of course considerably higher, or nearly in the proportion of six to one. The expenses for working an amount of material with 5 tons of solid contents are, for instance, on the old table about 12s., on the new one a trifle over 2s. The motive power required for one of these tables is very small indeed, considering the size of the machine: 1 H.P. is quite sufficient to turn 10 to 15 of them.

There is a further special advantage connected with the construction and mode of action of this rotating table, namely, that with some modification in the arrangement of the supply and clean-water troughs, &c., and some additions, two different classes of material can, if required, be worked separately on one and the same table. For this purpose, 2 of the 4 triangular distributing-planes have to be devoted to each class of stuff. The supply- and reception-troughs have to be properly partitioned off, and there are in addition required one special cleaning-trough and one pipe for washing off the ore (both intermediate between the two pair of triangular planes), also double the number of launders for supplying stuff and conveying the products to their respective settling-pits.

Buddles.—Buddles are made in considerable variety, but mostly for dressing the ores of the base metals in Europe, where they have long been used and whence they were copied for saving auriferous pyrites in gold mining countries. For this latter purpose, however, they are retreating in favour of modern improved concentrating machines to a large extent. The form chiefly retained for concentrating auriferous pyrites is that known as Munday's improved round buddle, shown in Fig. 230.

In this buddle, the sand enters the receiving trough fixed on the axle, and is thence conveyed through pipes to the rim of the basin, where it is

FIG 230



MUNLAY'S ROUND BUDDLE.

discharged; the heavier portion of the sand thus treated gradually settles down to the bottom of the basin, while the lighter portion washes away. The detention of the heavy sand is facilitated by the scrapers being fixed angularly on the arms and intercepting the sand as it flows down from the edge toward the centre, and causing it to return toward the rim. The detention of the heavy sand is likewise facilitated by the recesses formed by the circular ribs attached to the bottom of the basin.

The action of the scrapers is believed to be improved by arranging them in a spiral form, one succeeding the other at about distances of $1\frac{1}{2}$ in.

The heaviest portion of the material treated is found to accumulate within 2 ft. of the rim of the basin.

The circular basin may be made of wood, iron, or masonry, and of any convenient size from 12 up to 34 ft. diameter. Motive power may be communicated by belting or other means from existing machinery. The power to drive a 24-ft. buddle may be estimated at from 2 to 4 H.P.; the speed of the buddle is 5 revolutions a minute. Water from the stamps to be regulated according to the nature of the sand. A buddle of 24 ft. diameter will efficiently treat from 20 to 30 tons of sand a day, taking it from the batteries. A slope of 1 in. to 1 ft. is generally given to the bottom of the basin. The angle at which the scrapers are set with the arms is about 18° from the right angle.

This buddle has been successfully adopted at the Port Phillip Company's works, and the New North Clunes claim, at Clunes; at Llanberis, Ballarat; Koch's crushing plant, Sandhurst; the Walhalla claim at Stringer's Creek, and at several other mining claims in Victoria. But in some cases slight modifications are adopted. For instance, those used at the Port Phillip works are constructed of brickwork and cement instead of iron, and are then found to be cheaper while equally efficient. The ribs marked * in the plan are not used by Rosales at the Walhalla works; and the figures 1 to 7 show the position and angle at which the scrapers are set, Nos. 4 and 7 being 17 in. long, i. e., $1\frac{1}{2}$ in. longer in front and 2 in. longer at the back than the others. Rosales prefers this arrangement to Munday's. The scrapers used by Rosales are of india-rubber, and their shape is shown at B, Fig. 230.

Alve's Concentrator.—At the Sons of Freedom Mine, Boggy Creek, Victoria, one of Alve's patent concentrators has been at work for some time, and has given satisfactory results. The local mining registrar thus describes the apparatus:—

"There are three lengths of tables, each 6 ft. by 2 ft.; blankets on the tables, and wire screen $\frac{3}{8}$ in. mesh firmly fixed on blankets; the pyrites are caught in the wire-netting, the blankets are washed every hour, then taken and put through a reducing table of the same construction with a

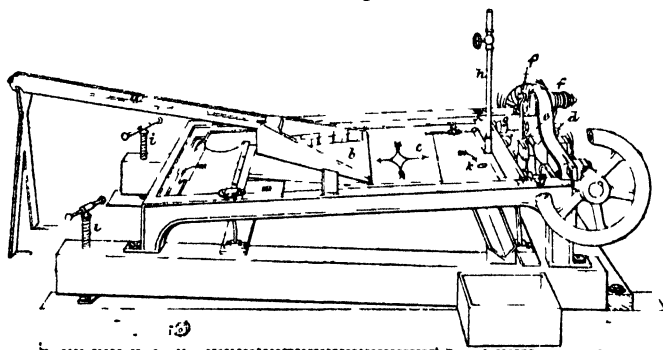
greater pitch and 6 ft. longer. After leaving the reducing tables, the pyrites are thoroughly clean, and are dried on an iron plate over a slow fire. The blanketing tables have a pitch of about 1 in. to the foot, and the reducing tables $1\frac{1}{2}$ in. to the foot. At the end of the blanketing tables there is an amalgamator, which is charged with about 70 lb. of mercury, and is so constructed that everything must come through the mercury; after which there are several ripples to catch any mercury that may escape, which is next to nothing. Previous to Alve's patent being attached to the Sons of Freedom battery, I am informed by Mr. Preston the mining manager, that the assay of the tailings, both at the School of Mines, Ballarat, and Sandhurst, was $6\frac{1}{2}$ dwt. per ton. Since the fixing of Alve's patent, no particles of gold can be found outside of the machine, and in the opinion of Mr. Preston, it is the most simple and efficient pyrites saving apparatus at the present time."

These machines are also in use at the Rob Roy and Hans companies and at Walhalla. It is said that by their operation tailings containing 3 gr. of gold per ton can be profitably treated, the machines being designed to reduce the cost of working to a minimum. No machinery in motion is required, the whole apparatus being worked by gravitation. It is light, portable and durable, and works with very little water pressure, catching in the amalgamator any floating gold there may be, and in the concentrator any auriferous grains of sand or pyrites. The cost of a machine capable of treating 100 tons of tailings per week is about £30.

Dodge's Concentrator.—The concentrator introduced by M. B. Dodge, and made by Malter, Lind, & Co., 189, Broadway, New York, is shown in Fig. 231. The pulp from the batteries flows down the sluice *a* on to the distributing-board *b*, which is provided with spreaders. The sluice and distributing-board are inclined in an opposite direction from the concentrating table *c*. The pulp is divided evenly across the table by the spreader at a point just below the double arrow. Under the spreading-board is a riffle, to arrest the downward movement of the mineral when it leaves the spreader. An end shake is given by the cam *d* against the lug or tappet *e*, moving the concentrating-table gently forward, when the springs *f* draw the table suddenly back, striking on the upper corners of the table against the buffers *g*, which causes the pulp to move upwards on the machine. The pipe *h* delivers clear water, through cross pipes provided with small orifices, in jets against the head of the table; the water flows downward in a thin sheet against the pulp, causing the gangue to move down and out of the table at the lower end, as shown by the arrow; while the mineral, being heavier than the gangue, moves upwards, as shown by the double arrow, against the clear water into the depression and through the orifice. This orifice is provided with a wooden plug, with a suitable passage for mineral, and water

enough to convey the concentrations down the short sluice into the box provided to receive them; thus forming a continuous discharge of mineral and gangue from the machine. At the lower end of the table is a receptacle to receive any fine mineral and mercury that might be left in the gangue while passing over it, which can be removed from time to time, or discharged continuously. This is one of the great advantages claimed for this concentrator over all others. The machine can be

FIG. 231.

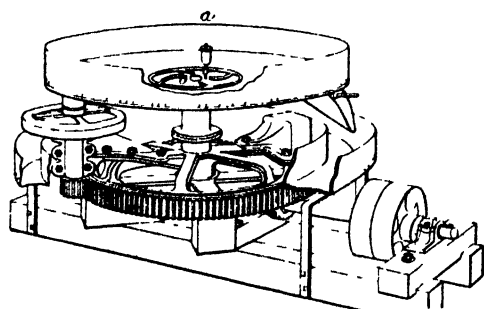


DODGE'S CONCENTRATOR.

regulated for different kinds or grades of ores, by the amount of clear water turned on, and by the length of stroke, which is regulated by jam-nuts on the buffers *g*; also, by means of the screws *i*, the machine may be raised or lowered to regulate the incline of the table. A great deal depends on the pressure put on the springs *f*.

Duncan Concentrator.—As will be seen from the illustration, Fig. 232, this machine is neither an imitation of, nor an improvement on any of

FIG. 232.



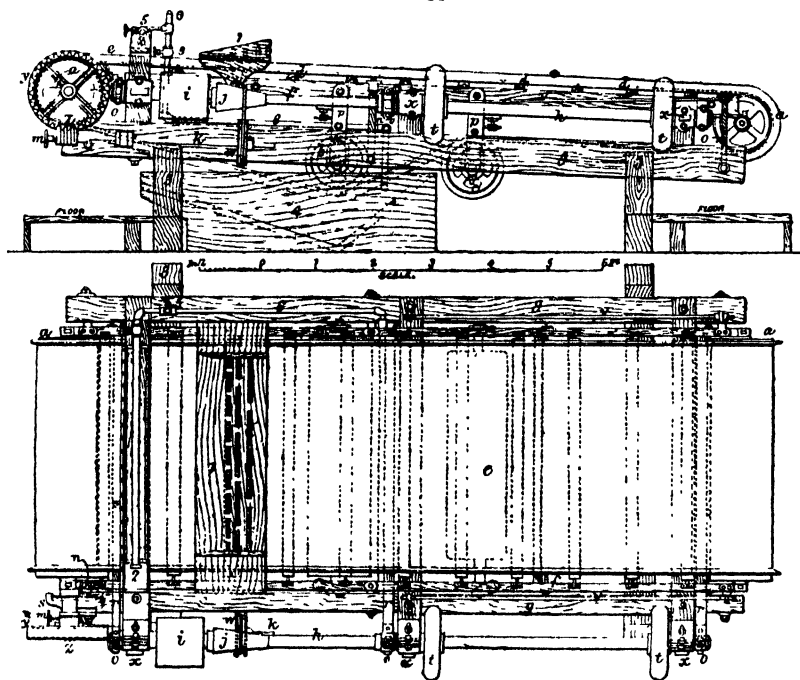
DUNCAN CONCENTRATOR.

the numerous belt machines or percussion tables; but by an ingenious arrangement of gearing, a combination of movements is obtained, closely resembling the motion given to the prospector's dish in washing and panning, and very similar to that of the vanning shovel. The revolving motion of the pan *a* causes the sulphides and other heavy minerals to flow towards the

circumference and settle to the bottom, the gangue being held in suspension and gradually carried away by the force of the current

down the central discharge *b*, whilst a transverse movement, obtained by an eccentric working on the underside of the pan, materially assists in throwing down the heavier portions of the pulp, and facilitates the speedy and complete separation of the ore from the gangue. Means of adjustment are provided, so that the movement of the pan can be modified to suit various kinds of ore; and argentiferous galena, copper pyrites, &c., are concentrated as successfully as gold or silver ores. It is claimed for this concentrator that it has fewer component parts than other machines; that of those parts only three or four are subject to any special amount of wear; that they can be replaced, when required, at a very moderate cost; that the first cost as well as the weight for transport is considerably less than other machines of equal capacity; and that it is erected more cheaply and quickly; whilst it also occupies less space in the mill. On some classes of mineral it has proved itself superior to all other concentrators tried in competition with it. The makers are Charles Appleby & Co., 89, Cannon Street, London.

FIG. 233.



FRUE VANNER.

Frue Vanner.—The Frue “vanner,” made by Fraser and Chalmers, 145, Fulton St., Chicago, Ill., and by Calvert, Cornes, & Harris, 30, Walbrook, London, is shown in Fig. 233. *a* are the main rollers that carry

the belt and form the ends of the table ; each roller is 50 in. long and 13 in. in diameter. In order that they may be light, yet strong and durable, these rollers are made of No. 16 sheet-iron, riveted lengthways, and crowned in the centre about $\frac{1}{4}$ in. The roller is secured at the ends by rivets to light cast-iron frames. The whole is galvanised when finished, so that even the rivets are protected from rust. The roller when finished is strong, and only weighs 70 lb. The bolts which fasten the boxes of *a* to the ends of *f*, also fasten to *f* the chilled cast-iron supports of the flat bars of iron *n*. *b* and *c* are of the same diameter, and are made in the same way as *a*. The belt *e* passes through water underneath *b*, depositing its concentrates in the box 4 ; then, passing out of the water, the belt *e* passes over the tightening-roller *c*. *b* and *c* are hung to the shaking-frame *f* by straps *p*, which swing on the bolts fastening them to *f*. By means of the hand-wheels, *b* and *c* can be swung on either side, thus tightening and also controlling the belt.

The boxes holding *a* in place have slots, so that by drawing out or shortening, *a* can be employed as a means of tightening the belt *e* ; and as *e* sometimes travels too much toward one side, this tendency can be stopped most quickly by lengthening or shortening on one end or the other of *a*. The swinging of *b* and *c* out of line also controls the belt, but neither has influence equal to *a*. The small galvanised iron rollers *d* and their support cause the belt *e* to form the surface of the evenly inclined plane table. This movable and shaking table has a frame *f* of ash, bolted together, and with *a* and *a* as its extremities. This frame is braced by 5 cross-pieces (shown by dotted lines). The bolts holding together the frames pass through the sides close to the cross-pieces ; the cross-pieces are parallel with *a* and *d*, and their position can be understood by the 3 flat spring connections *ro*, which are bolted to 3 of them, one to each, underneath the frame. The belt *e* is 4 ft. wide, $27\frac{1}{2}$ ft. in entire length—being an endless belt of rubber with raised sides.

The stationary frame *g* is bound together by 3 cross-timbers, which are extended on one side to support the crank-shaft *h*. *g* supports the whole machine, and the grade or inclination of the belt is given by elevating or depressing the lower end of *g*. This is accomplished by means of wedges ; for this frame rests on uprights 3, fastened to two sills, which form the foundation of the machines in the mill. *f* is supported on *g* by uprights *n*, 3 on each side. These uprights are of flat wrought iron, drawn to a knife-edge at each end, and case-hardened, with bearings above and below of chilled cast iron ; each middle bearing on *f* has one bolt-hole, and there are two of them, one on each side. The end ones have two bolt-holes, and there are four of them, two on each side. These bolts pass through the frame *f*, and hold to the frame the bearings of *a*, which work in a slot. The bearings of the head-

roller are higher than those of the foot-roller; i. e. a is a trifle higher than the regular plane of the table, and the first small roller d should be raised a little.

The cross-timbers binding together g and resting on them are extended on one side, and on these extensions rest, with its connections, the main or crank-shaft h , in bearings x ; the cranks are $\frac{1}{2}$ in. out of centre, thus giving 1 in. throw. The driving-pulley i forms with its belt the entire connection with the power. j is a cone-pulley on the crank-shaft h . By shifting the small leather belt, the uphill travel of the main belt e is increased or diminished at will. The small belt connects to j the grooved pulley w , which is on the small shaft k , and by means of the hand wheel can be shifted on k and held in place. The two bearings of k are fastened to the swing-box y , a cast-iron shell protecting the worm z and worm-gear l ; y turns on a bearing bolted to the outside of g , and thus becomes a swing-box for swinging w and k . The object gained by this is that the weight of w and k (swinging with y) hangs on the small leather belt, and keeps it tight, so that this small belt will last for a year without slipping or breaking. Before this improvement, the small belt was constantly breaking or slipping. In some cases, this movement is accomplished with step pulleys and flat belt. A hand-wheel m is used to relieve the small belt from part of the weight of k and w ; by screwing it up, k and w can also be raised, taking all the strain off the small belt, and thus stopping the uphill travel. k terminates in a worm z , which connects with a worm-gear l , travelling in a bearing bolted to the outside of g . z and l are protected from dirt and water by the cast-iron shell y enveloping both.

The short shaft which l revolves terminates in an arm s , which drives a flat steel spring q (which is a section of a circle), connected with the gudgeon of a . r are 3 flat steel spring connections bolted underneath the cross-pieces of f , and attached to the cranks of the shaft h by brass boxes o . These springs give the quick lateral motion—about 200 a minute. t are two fly-wheels. v are two rods passing from the middle cross-timber to the upper bearings of the lower uprights n . The cast-iron washers on the bolts of the cross-timber have lugs cast on them, and so have the bearings of the lower n . v pass through these lugs, and at each end are nuts on each side of the lugs. Thus v prevent the whole movable frame f from sliding either up or down, and by them f is squared. 2 is the clear-water distributor, and is a wooden trough, which is supplied with water by a perforated pipe; the water discharges on the belt in drops by grooves $1\frac{1}{4}$ in. apart. 1 is the ore-spreader, which moves with f , and delivers the ore and water evenly on the belt. 3 are upright posts, which are firmly fastened to two sills, forming the foundation for any number of machines. 4 is the concentrates-box, in which the water

is kept at the right height to wash the surface of the belt as it passes through. 5 are the cocks to regulate the water from the pipes 6.

The ore is fed with water on the belt *e* by means of the spreader 1. Thus the feed is uniform across the belt. A small amount of clear water is distributed by 2. A depth of $\frac{1}{2}$ in. of sand and water is constantly kept on the table, and the table should receive about 200 shakes a minute. The uphill travel or progressive motion varies from 3 ft. to 12 ft. a minute, according to the ore; and the grade or inclination of the table is from 4 to 12 in. in 12 ft., varying with the ore. As previously explained, the inclination can be changed at will by wedges at the foot of the machine, these wedges being under the lower end of *g*, and resting on uprights from the main timber of the mill. The amount of water used, the grade, and the uphill travel, must be regulated for every ore individually; but once established, no further trouble will be experienced in the manipulation. In setting up the machine, everything must be in line, except the tightener-roller *c*. The tightener-roller not only tightens the belt but regulates it and keeps it in place on the table. This wide belt travels uphill very slowly, so that it takes several minutes to recover its central position on the table, and at times one bearing may necessarily be several inches farther up than the bearing on the opposite side, thus twisting *c* out of line. In treating ore directly from the stamp, too much water may possibly be used by the stamps for proper treatment of the sand by the machine. In such a case, there should be a box between the stamps and the concentrator, from which the sand with the proper amount of water can be drawn from the bottom, and the superfluous water will pass away from the top of the box; but as mineral will also pass away with this water, there should be settling-tanks for this water, and the settlings can be worked from time to time as they accumulate.

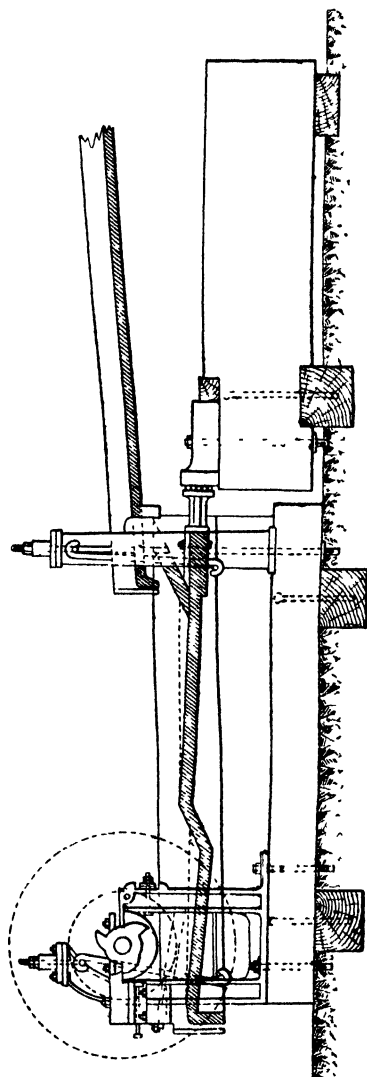
The surface of the belt lasts for 3 months at least. As soon as the belt shows wear, it should be preserved; and the belt should never be allowed to wear to the canvas. The belt is preserved by a coat of rubber paint, prepared expressly for the purpose. Accidental breaks in the belt may be repaired by rubber cement. In renewing the surface of the belt, it must be dry. The paint should be thinned, if necessary, with benzine and naphtha, so that it readily flows from the painter's brush. The surface of the belt should be cleaned with naphtha. Then, a man standing at the lower end *a* paints liberally across the belt, and for 2 or 3 ft. up, as he can conveniently reach; then revolves the belt in its usual direction, i. e. upwards, for a short distance, and paints another short piece. This operation is continued until the whole surface has been painted. The rubber paint dries almost immediately, and in a very short time the belt is ready for work again. Every two months this

should be repeated. The paint should be put on uniformly, but not so hastily but that the portions painted have time to become nearly dry before reaching the tightener *c*. In using the paint, it must be kept well stirred. The main body of the belt suffers hardly any wear, since it merely drags its own weight slowly around the freely revolving rollers ; and the life of the belt is lengthened by this precaution, viz., to keep it clean from sand at every point except the working surface, thus sand cannot come between the belt and the various rollers.

The concentrates-box 4, which is kept full of water, and through which *e* passes, may be of any size or depth desired ; in front of it may be an apron to catch any chance droppings of concentrates from the belt. Though not indispensable, it is best to have a few jets of water playing above and underneath on the belt as it emerges from the water in 4, so as to wash back any fine material adhering to the belt, and as such a method will cause an overflow in 4, the waste water, being full of finely divided mineral, should be settled carefully in a box outside. Every few hours the concentrates may be scraped out with a hoe, into a small box that can be placed under the inclined end of 4, and if this box be on wheels it can be readily run on a track to the place where the concentrates are stored : such a method seems clumsy, but there is comparatively a small quantity to handle. Frequently the sand on the belt forms a corner on each side, and to break up these corners

and keep a uniform consistency on the belt, a system of drops or small jets of water can be used on each side to advantage. Such will help to increase the capacity of the machine, and will enable it to do uniformly better

FIG. 234.



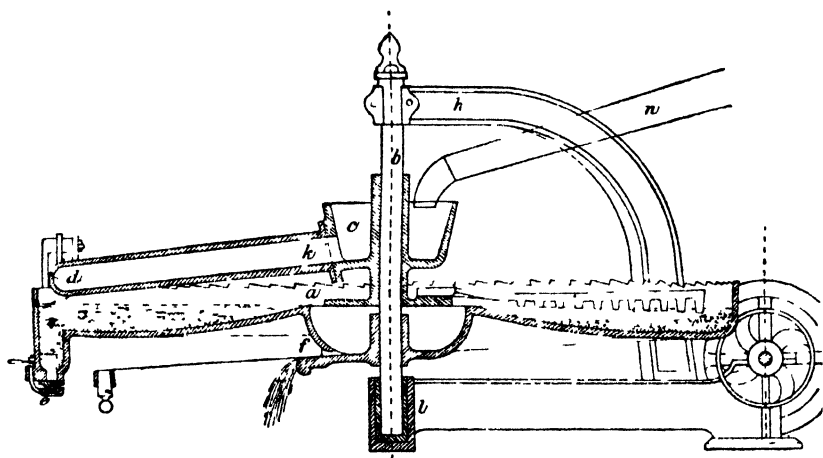
HALLEY'S PERCUSSION TABLE.

work. The latest improvement is to corrugate the large travelling belt; recent trials of this modification have been very satisfactory.

Halley's Percussion Table.—This machine is shown in Fig. 234. An inclined table about 8 ft. by 4 ft. is slung at the corners, the inclination being adjustable. A 3-toothed cam rotating against a spring gives a reciprocating motion. The ore to be classified is fed on at the upper end with water. By the repeated concussions the light and heavy ingredients are separated, the former flowing away with the water, while the latter are retained in the depressions of the table. Each 5 head of stamps requires one table.

Hendy's Concentrator.—This apparatus, Fig. 235, consists of a shallow iron pan, 5 to 6 ft. in diameter, supported by a vertical shaft in the centre, and made to oscillate to and fro by means of cranks on a shaft at

FIG. 235.



HENDY'S CONCENTRATOR.

one side, and joined by connecting-rods to the periphery of the pan. The pan turns for a short distance at every revolution of the crank-shaft. A frame supports the central pin and crank-shaft, as well as arched arms *h*, which rise over the pan and sustain the upper end of the vertical shaft *b*. The bottom of the pan is raised in the centre around the shaft, nearly to the height of the rim, and thence descends towards the periphery in a parabolic curve, by which the movement of the particles from the centre towards the circumference is facilitated, and their passage in the other direction obstructed.

When placed for operation, the apparatus should be perfectly level. The stuff to be concentrated is delivered by the trough *n* to the hopper *c*, whence it is fed through the pipe *k* and distributor *d* into the pan near

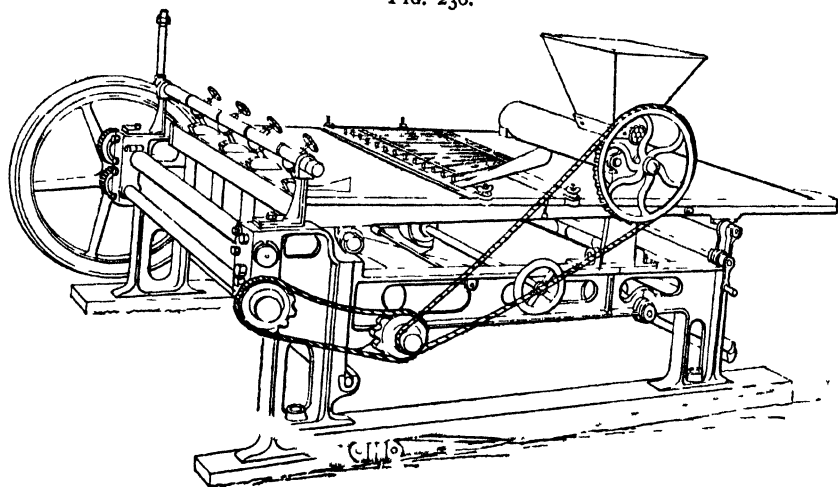
its outer edge. The feeding extends around the whole circumference, by causing the distributor to rotate around the vertical shaft, accomplished by the movement of the pan. The upper edge of the pan is a continuous ratchet, into which 2 pawls connected with *d* drop during the motion of the pan from the distributor, and in the return motion give a velocity to the distributor equal to that of the pan. Continued impulses in this way keep the distributor in regular rotation around the shaft. Rake-like arms are bolted to a flange on the bottom of the hopper *c*, and are carried around with the distributor, serving to separate the compact mass of sand and pyrites as it settles, and breaking the scum that gathers on the surface. The crank-shaft makes 200 to 220 revolutions a minute, thus throwing the pan to and fro an equal number of times, and keeping the materials in a constant state of agitation. The heavier substances, such as the pyrites and any stray particles of mercury or amalgam, settle to the bottom, and accumulate in the lowest parts of the pan, gradually displacing the sand and lighter materials, which, with the excess of water, flow over the raised bottom at the centre, and out of the pan by a central discharge. The accumulated sulphurets are discharged at the gate *e*, the opening of which is regulated by a small handle at the front of the machine. The pyrites that are discharged may be received into boxes or troughs.

These machines weigh about 1000 lb. each. They are run by a belt, and usually set in pairs. The amount of water required is not large: not more than what flows away from the batteries with the sands to be concentrated. Each machine will receive and concentrate 5 tons of stuff every 24 hours; 8 tons have been put through in that time, but the product was not entirely free from sand, the presence of which is not objectionable in some processes of working, and if clean pyrites are desired, the discharge from 4 machines is delivered into a fifth, and this gives a complete clean concentration.

Imlay Concentrator.—Favourable accounts are given of the Imlay concentrator and amalgamator, shown in Fig. 236. Described generally, the machine consists of a flat table having a copper surface, with 2 up-turned sides and one similar end, the opposite end being open to permit the discharge of the tailings. This table is set at an incline, varying from $\frac{1}{2}$ to 2 in., the waste discharge end being lowest. At its opposite upper end, the table is provided with outlets for the reception and discharge of the concentrations. The table is supported upon 4 arms, one at or about each corner, which arms project upwardly from two transverse rock shafts at either end of the machine and about 1 ft. below the table; when motion is duly communicated, these arms vibrate to and fro, a longitudinal reciprocating movement, or a lengthwise movement, of the tables being thus effected.

If this to and fro movement were in equal time, the results would not be those now obtained. Hence, a variable movement, which is the peculiar characteristic of this machine, is obtained by a very ingenious

FIG. 236.



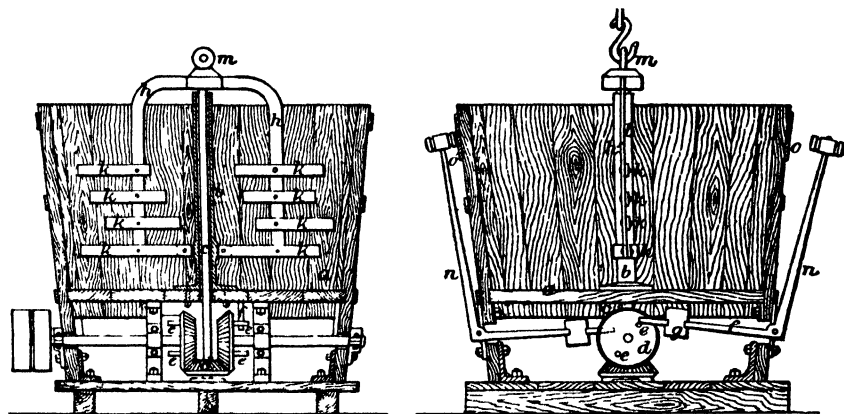
IMLAY CONCLNTRATOR.

and yet simple combination of mechanism. The main shaft of the machine is provided with an eccentric gear-wheel, which meshes with a like gear on a counter-shaft, parallel with the main shaft. This counter-shaft is an eccentric or crank-shaft, from which two connecting-rods lead to the reciprocating-table. The eccentric gears communicate a variable rotary movement to the crank-shaft, and this, through the connecting-rods, produces the variable reciprocating lengthwise movement of the table. The effect of this movement is to cause any material above a given specific gravity, laid upon or fed to the table, to travel upwardly upon the latter, while anything below such gravity will be caused to pass down the same. As the pulp almost invariably carries some mercury, the latter soon forms an amalgamated surface on the copper-plates.

Keeve, or Tossing-tub.—This is shown in Fig. 237, and measures 30 in. deep, 4 ft. wide at top, tapering to 3½ ft. at bottom; it consists of 2-in. wooden staves, bound with 3 iron hoops, 2½ in. wide and ¼ in. thick. The staves are carried 2 in. below the floor of the tub, and 4 of them are prolonged a further 10 in., and are bolted to a platform 6 in. high, so that the floor of the tub is 18 in. above the floor of the mill. A hollow cast-iron tube *b*, 3 in. in diameter, passes centrally through the floor of the tub, and is bolted to it by a flange cast on the tube. This tube is traversed by a 2-in. wrought-iron shaft *c*, carrying a yoke *h*, and eye *m*, to which 8 flat iron stirrers *k*, 2 in. wide, are bolted. The shaft is rotated by gear-wheels *d*, to which are attached 2 pins *e*, which catch

an arm *f*, bearing counterpoises *g*, which end in hammers *n*. When the tub is to be charged, a wooden wedge is pushed between the side of the tub and the hammer handle, to prevent the pins *e* striking the lever *f*. First the tub is half filled with water, and the stirrers are rotated 48 times a minute; then the tub is almost filled with tailings, which should be introduced near the outer edge. This accomplished, the stirrers are stopped and lifted out by means of the hook and rope attached to *m*, and the

FIG 237.



KEEVE OR TOSsing-TUB.

wedge is withdrawn so that the hammers can play on the sides of the tub at *o*. When this has proceeded long enough to completely settle the contents of the tub, the water is drawn off at the top, and the settlings are divided into "tops," "middles," and "bottoms." The tops, extending about 2 in. deep, are scraped off and discarded. The middles are re-tossed, and the tops from these again discarded. The remainder of the middles goes back to be buddled, while the bottoms should consist entirely of clean sulphurets.

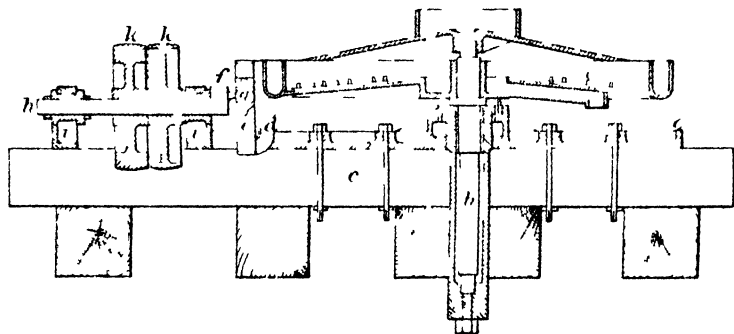
McNeill's Concentrator.—Mr. J. R. McNeill, engineer of the Long Tunnel Gold Mining Company, Walhalla, Victoria, has made some important improvements on what are known as Brown and Stansfield's concentrating pans. These pans were formerly used at the Long Tunnel battery, but were discarded owing to the amount of wear and tear and breakages. McNeill's improvements consist in minimising the wear and tear by a different method of oscillation and suspension of the pans, but principally in lining the pans with mercury-coated copper plates, which are thickly studded with copper pins similarly coated. The materials treated by the Long Tunnel Company are pyrites and blanketings caught from the tailings after having passed through

the different gold-saving appliances inside the battery, and the gold obtained is, therefore, so much absolutely saved from going into the waste heap. Mr. Ramsay Thomson, the manager of the company, saved, in 3 months, by means of the improved pans, 66 oz. 5 dwt. 8 gr. of gold, which would otherwise have been lost.

Mr. William Parker, of the Long Tunnel Extended Company, has also used McNeill's pans, and in reply to inquiries he wrote :—"I have no hesitation in saying that McNeill's is the best concentrator that I have yet seen, as it is not only a good separator of the pyrites from the quartz-sand, but it is also a good amalgamator, and, if worked judiciously, is capable of bringing together a very large proportion of the broken-up floured mercury, and forming it into amalgam. I consider this one of the most important features in connection with the machine, because simple concentration is not much trouble, but the collecting of the floured mercury in the form of amalgam is, and up to the present time has baffled the efforts of our best men."

The following is a description of the machine :—In Fig. 238, *a* is the concentrator pan, the working surface of which is lined with mercury-coated copper plates ; *b*, vertical spindle ; and *c*, supporting frame. *d* are brackets formed upon or secured to the outside of the pan, having bolted to them the flat bars *e*, between which the sliding block *f* works. Such

FIG 238



MCNEILL'S CONCENTRATOR

sliding block is made in halves, bolted together at its end snugs, and bored to fit the ball crank-pin *g*, which is secured or formed upon the end of the shaft *h*, which is supported in bearings *i* in the framing, and upon which shaft is fixed the driving-pulleys *k*. The concentrator pans have snugs formed on their bottoms, in which are secured ball pins, to which the ends of a connecting rod are attached, while motion is imparted to the connecting rod at its centre, where a slot eccentric is formed for a

sliding block, such slot eccentric being formed by the two T ends of the connecting rod being bolted together with a distance ferrule between them, the sliding block in this case working upon a plain crank pin on the end of the crank shaft. Stays are attached to the slides of the pan, and at their upper ends meet at a common centre, which is supported in the overhead framing.

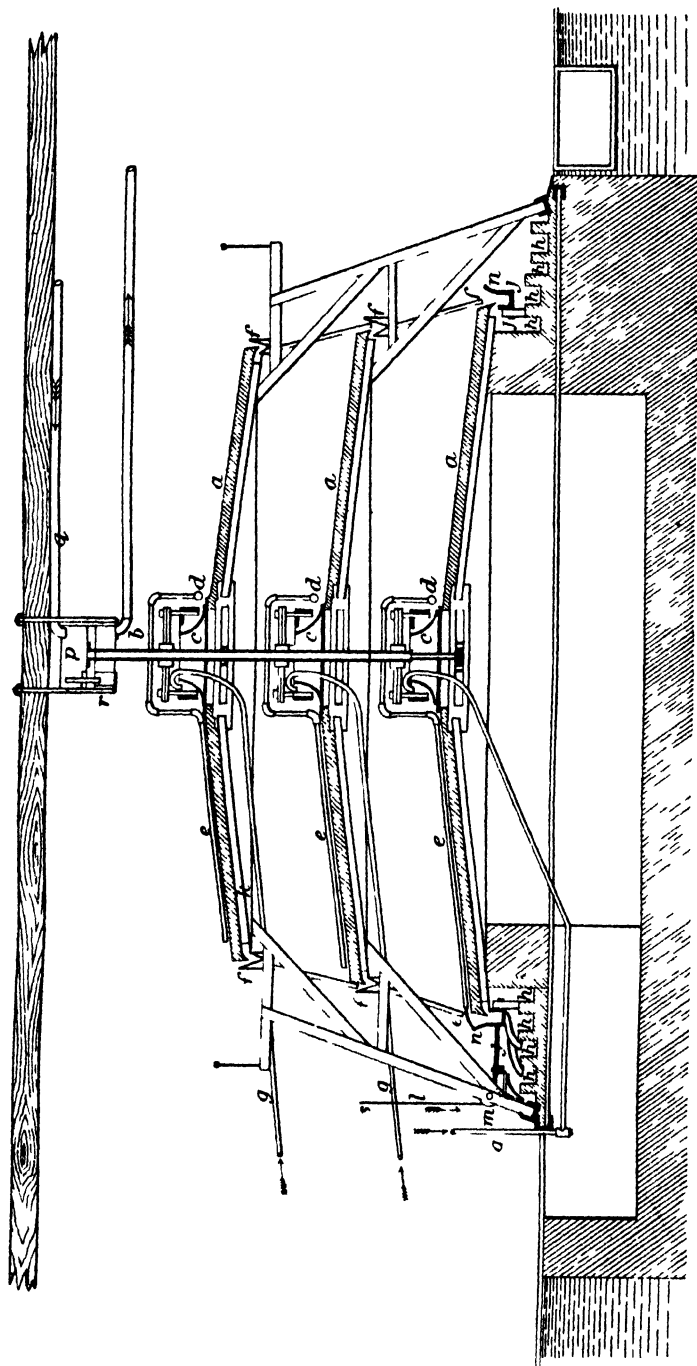
Rew and Jones's Concentrator.—This has a reciprocating motion, and is designed on the principle of the ordinary miner's pan. It is claimed to be successful even with the finest sulphurets.

Stationary Slime Table.—In Fig. 239 is shown a treble stationary slime table with rotary feeder and cleaner, from which greatly improved results are expected.

It is unnecessary to repeat that the greatest difficulties and comparatively heaviest losses in ore dressing are experienced in the extraction of the ore from the slimes. The appliances generally used have been described in the preceding pages. With buddles, the ores are deposited on the table in concentric rings, the heaviest near the centre, the lightest near the periphery, and the barren sand has to be swept from the table by the water. The operation of separating the different kinds of ore and seconds must be executed by hand, by digging out separately the concentric rings. The separation is done in the direction of the radius, and it is evident that, although the buddle can be made of very large diameter (say 30 ft.), there is no distinct classification possible, as the various grades are mixed at their boundary. Moreover, the proper execution of the separation rests entirely upon the ability of the workmen; the surface of the table, when covered with a quantity of ore-slimes, becomes irregular, and the most important condition for good working of this apparatus, viz., a smooth sheet of water running down the table-surface, is not complied with any longer. It must be pointed out that in case the difference of specific gravities of ores and quartz is used for separation, the stuff which is to be treated must be properly prepared. The classification to size of slimes by aid of sieves is impossible; and, in fact, there is no practical means known to execute such classification. Rittinger invented his "Spitzkasten" (pyramidal boxes with a syphon-outlet at the bottom) with a view that, if a current of slimes passes the box, those particles which have the greatest tendency to settle will (in consequence of the reduced velocity of the current in the box) fall to the bottom and pass out sideways, whilst the lighter particles are carried away with the current. And if a series of boxes increasing in size are placed one behind the other, the velocity of the current will decrease, and the bottom-efflux of each following box will consist of particles of less tendency to settle—i. e. of finer grains.

But although the bottom-efflux from the first box will be coarser than

FIG. 239.



STATIONARY SLIVE TABLE.

that of any one of the following, and that of the last (and largest) box will be the finest, it cannot be denied that the grains of the efflux of any one of them cannot be of uniform size, except all grains have the same specific gravity, which is contrary to experience. If the slime which is to be treated contains galena, pyrites, and quartz, it is evident that the efflux from the bottom of any box must show the above minerals in sizes proportionate to the specific gravity, viz., large quartz-grains, medium pyrites-grains, and small galena-grains, and it is impossible to separate them in a jigger. In Germany such grains are called "gleichfällig," which means grains of equal falling speed in water or air; and it is clear that such grains cannot be separated in jiggers which are making use of the difference in the "falling-speeds" of the grains, which must be of uniform size and various specific gravity.

Buddles and those tables which are used for the separation of slimes must be based on a different principle, if a perfect separation is to be obtained. The stuff treated on them, viz., the bottom-efflux of one Spitzkasten (at the time) consists of grains of equal falling-speed, but various sizes; the largest grains being those of lowest specific gravity, and, consequently, offering to a horizontal or nearly horizontal current of water not only the largest surface, but also the smallest resistance; and if these large light grains are exposed, together with small heavy grains (of equal absolute weight), to a current of water running over an inclined smooth surface, the large light grains will advance quicker than the others; and if the current is properly regulated, the large grains will run off the incline with the water, whilst the heavy grains are deposited on the table, the heaviest first. This operation requires, of course, a perfectly smooth surface, and all appliances constructed with the intention of retaining the clean ore on the surface can only work effectively during the first rotations of the feeder. As soon as the surface is covered or partially covered with ore, the action becomes irregular.

This was the reason why buddles were replaced by rotary conical tables with fixed feeders and cleaners. In the course of a rotation, a sector of the conical surface first passes a feeder (fixed near the centre) where it receives a certain quantity of slime: it then passes a spray of clean water, ejected from the centre, and strong enough to carry down the quartz and lighter ore-grains whilst the specific heaviest grains are settling. After that, the said sector passes underneath a water-basin fixed at a certain distance from the centre and ejecting a light spray on to the table, sufficient to sweep down the quartz grains only. At a third portion of the table similar arrangements can be made for washing down the ores of lighter specific gravity deposited near the periphery; and before the sector of the table (under consideration) comes again to the fixed feeder, the heavy ore is brushed off. Of course, separate gutters

are fixed under the drip, so as to receive the various kinds of ores, seconds, and tailings. The said sector then appears again in front of the feeder perfectly cleaned.

It follows from the above description, that with a buddle, the separation is executed by grouping the various kinds of minerals in concentric rings, and that with the rotary table the different kinds are obtained on the periphery. As a matter of fact, in neither of these cases are the various kinds of minerals sharply divided; if galena is the heaviest, there will be a space on the table where it is mixed with pyrites, and it is easily understood that the dimensions of the table are of great influence. On a small table or buddle, it will scarcely be possible to divide the pyrites from the zinc-blende, and, consequently, large diameters are necessary for both apparatus, and this is somewhat difficult to execute with the rotary tables. If their diameter exceeds 15 ft., they begin to oscillate, and during a rotation the incline of the table is always changing.

Notwithstanding the superiority of rotary tables as separators over buddles, which are in fact concentrators only, the said defect could not be overlooked, and the inventor of the stationary table succeeded in eliminating this defect, and combining the good qualities of both buddle and rotary tables, by fixing the table and making feeder and cleaners rotating. Any diameter can now be used, a perfectly smooth surface is (polished cement) obtained, and the separation of the various kinds of mineral is carried out to perfection.

On the old rotary table, the gutter which surrounds it under the dripping-nose is divided into as many compartments as various kinds of stuff are to be obtained, and each compartment is connected by a pipe with a settling pit. The places of discharge of the various kinds of stuff are, in this case, fixed, as they are at certain distances from the fixed feeders and cleaning-pipe. But as, on the new table, the latter are rotating, it is necessary that the said gutter is also rotating, and that as many stationary ring-shaped gutters are surrounding the table as kinds of stuff are to be obtained. The rotating gutter, which, of course, is divided into the same number of compartments, is provided in each of the latter with a spout leading into one of the ring-shaped stationary gutters. Each of the latter is in communication with a settling pit by aid of a clay pipe.

The arrangement is shown in Fig. 239: *n* is the rotating gutter of sheet iron fitted with wheels, which are running on a circular rail; *j* are the spouts leading into the fixed gutters *h*; *g* shows the feeding-pipe leading from a "Spitzkasten" to the feeding-basin *c*; only a part of the latter allows the efflux of the slime on the table surface *a*; the spindle *b* is hollow, and supplies the pipes *d* and *e* with water for cleaning the table;

the pipes *e* are fixed to the rotary gutter *n*, and the rotating motion is given to the latter, the spindle, the cleaning-pipes, and the feeding-basin *c*, by an endless chain slung round the gutter *n* and passing over the two pulleys *m* to a shaft under the ceiling, which is driven again by belt.

The sprays of water for washing the slime on the table are ejected from perforated pipes, which are movable and can be placed in any position required thereby; and as the table is thoroughly cleaned after each rotation, and its action can be easily controlled, a common labourer is able to regulate the water sprays in such manner as is required for perfect separation of the various classes.

The stationary table is more expensive than a buddle; but finally it is considerably cheaper, because it is a true separator, and the stuff which has been treated on it does not require further treatment; and it works without interruption, whilst buddles work intermittently, because they must be cleaned by hand. During this time, of course, the buddle cannot work, and, besides, wages are consumed for removing the deposited ores from the table. The stationary tables are made in three sizes, viz., of 19 ft. 8 in., 23 ft., and 26 ft. 3 in. diameter respectively. A table of 26 ft. 3 in. diameter is able to perfectly separate 7 tons of slime, dry weight, in 10 hours, and requires about 20 gal. of water per minute. The price of the iron parts of a stationary table of 26 ft. 3 in. diameter is, f.o.b. Australian ports, 280*l.*, including patent licence. The cost of the brickwork is (in Germany) about 75*l.*

WORKING RESULTS.

This chapter has perforce contained so many digressions interfering with the thread of the subject, that it will be interesting to embrace some of the leading facts in a few paragraphs dealing with generalities rather than details.

Richness maintained in depth.—In reviewing the progress of mining operations in the Central Division of Ballarat for a recent quarter-year, Mining Registrar D. Christy remarks that "it is satisfactory to be in a position to state that, as we proceed in quartz development, the theory of quartz-mining being unremunerative at deep levels is still further being exploded. How such a theory first obtained a footing, or on what premises it rested, is difficult to divine; but the practical experience of this and other gold-fields leads very forcibly to an opposite conclusion. The Band and Albion Consols Company for the week past crushed 283 tons of stone from a level of 1200 to 1300 ft., yielding 340 oz. of bar gold, an average of 1 oz. 4 dwt. to the ton. This steady

increase, leaving out the rich patches the company has obtained from time to time, inspires confidence in deep sinking. The Sulieman Pasha Company, in Ballarat East, have been well rewarded for proceeding to a little further depth in their workings, having obtained 203 oz. of gold from 70 tons of stone, an average of 2 oz. 11 dwt. to the ton." These facts suffice to confute any amount of theory, and though equally good results may not always follow deep sinking, there is a fair prospect of their doing so, and an inducement to at least prospect. Many a mine now closed as "exhausted" only awaits fresh prospecting.

Yields.—The yields of quartz veins vary of course to an unlimited degree, and no statement could show the possible maximum or minimum; but it will be of some interest to note what are the common experiences of yields over large districts and covering millions of tons raised and treated. For output of quartz ores (free milling stone as distinguished from complex ores) Australia is immeasurably foremost, and among Australian colonies Victoria takes the lead.

The average yield of over 9 million tons of quartz raised in Victoria in the decade 1878–1887 was just about 9 dwt. a ton; and from the 23½ million tons raised from the commencement of mining in the colony till the end of 1887 was 10 dwt. 11 gr., yet the value of the gold thus won was some 50 million pounds sterling. The ¾ million tons brought to bank in 1887 gave an average of 9 dwt. 10 gr. a ton. The greatest extremes of yield were a minimum of only 22 gr. per ton from 2860 tons raised in the Beechworth District, and a maximum of 4 oz. 16 dwt. for 2 tons mined in the Maryborough District. Classifying the output according to returns in oz. and dwt. it will be found that—

3,000 tons afforded less than	1 dwt.	per	pn.
3,000 " " more	2	"	1 "
45,000 " " "	3	"	" "
6,000 " " "	4	"	" "
42,500 " " "	5	"	" "
60,000 " " "	6	"	" "
250,000 " " "	7	"	" "
800 " " "	8	"	" "
117,000 " " "	9	"	" "
130,000 " " "	10	"	" "
18,000 " " "	11	"	" "
14,000 " " "	14	"	" "
1,250 " " "	15	"	" "
21,000 " " "	16	"	" "
700 " " "	17	"	" "
40,000 " " "	18	"	" "
1,100 " " "	19	"	" "
19,500 " " "	1 oz.	"	" "
3,000 " " "	1½	"	" "
850 " " "	2	"	" "
2 " " "	4	"	" "

To summarise these figures, it appears that something over a 14th part of the total (viz. 57,000 tons) did not produce an average of $\frac{1}{4}$ oz. per ton, and that nearly two-thirds (or 527,000 tons) did not yield $\frac{1}{2}$ oz. per ton. This means that over half a million tons of quartz are raised annually, in a country where wages are three times as high as in England, for a value of about 2*l.* a ton, this value sufficing to repay all cost of mining and milling and leave enough profit to make the industry worth following.

Cost and Profit.—Circumstances vary so widely at different mines that no general estimate of cost or profit could possibly be deduced in a form that would have any practical value ; but it may be useful to quote the prices paid for crushing quartz in Victoria. These range from a minimum of 4*s.* to a maximum of 15*s.* a ton, the majority being about 5*s.* a ton where water power is available, to 8*s.* and upwards where the motor is steam. The success of a quartz mine depends quite as much on favourable working conditions as on richness in gold. Hence it may happen that a mine carrying 5 oz. of gold to the ton, but badly situated, may be inferior as an investment to another showing only 5 dwt., but favourably circumstanced. There are a great many mines where 3 to 4 dwt. of gold cover every item of expenditure, the excess being clear profit. In fact, one Victorian mine, in the Blackwood Division, satisfied its owners with a yield of $1\frac{1}{2}$ –2 dwt. a ton, crushing with water power. On the other hand, mines which have been credited with giving extraordinary assay values have not paid for working. So much depends on local conditions and proper management.

Water Consumption.—The employment of a sufficient quantity of water in the battery is a point which is very apt to be neglected when supplies are limited. It would often be better to sacrifice the output by stopping some of the stamps, and appropriating the water thus liberated, rather than maintain the full crushing power with a diminished water inlet. This is especially the case where the material is soft and argillaceous, because it will tend to allow the slimy tailings to accumulate in the mercury troughs and on the amalgamated plates, whereby the gold will be prevented from making close contact with the mercury, and may be even carried away by the strength of the narrow streams thus created, besides which the mercury is liable to be splashed out of the troughs when the accumulated sludge is disturbed.

Hindrances to Amalgamation.—(a) Dirty Water.—It is impossible to be too careful to ensure that the water used in the battery is free from grease and from mineral salts in solution. Some experiments were conducted by Skey, in New Zealand, to ascertain the relative effects of using rain water, sea water, and mineralised water (the last-named being prepared by dissolving 6 gr. crystallised iron protosulphate

in 1 gal. water, to resemble ordinary mine-water). The results were that the

Rain water yielded	..	..	..	8 dwt. 17 gr. gold per ton.
Sea water	„	..	..	8 „ 14 „ „ „ „
Mine water	„	..	..	6 „ 1 „ „ „ „

so that the impure mine water gave over 31 per cent. less gold than the pure rain water. This in itself is highly significant, and when it is borne in mind that much veinstuff contains sulphate of iron, ready to be dissolved in the battery water, it is still more evident that the battery water should be of the best, in order somewhat to dilute and counteract the iron sulphate derived from the ore under treatment. It is necessary to provide against grease derived from lubricants used in the machinery getting into the battery water. A small quantity of sulphuretted hydrogen or sulph-hydrate of ammonia will create a grease-like film on the gold grains, which is as injurious as grease itself in hindering amalgamation.

(b) Dirty Gold.—The action of the stamp battery is attended by two drawbacks. In the first place, owing to the high malleability of gold in comparison with its natural rock associates, repeated hammering produces a certain amount of extremely thin tiny plates, which are sometimes so fine and light as actually to float for a considerable distance on the flowing water of the battery. This is one cause of loss, because the floating particles do not come in contact with the mercury. In the second place, it has been proved to demonstration that the hammered surface of a particle of gold is much less readily amalgamable than a surface which has not been beaten. This is so even when the surface retains its original purity and has not been contaminated with foreign matter. But owing to the softness of the metal, it cannot endure hammering in the presence of quartz powder without a certain amount of that powder becoming firmly attached to the beaten surfaces, and if the hammering and rubbing are prolonged sufficiently, the whole superficies of the gold fragment will be studded with very minute quartz grains. This coating may be, and often is, so complete and continuous that the mercury is altogether prevented from exerting its action upon the encased gold. This evil is likely to be greater with rotating stamps than square ones, and will probably increase with the weight of the stamp. Weeks of contact between the coated gold and ordinary mercury will not bring about amalgamation. But the galvanic action set up by contact with iron in solution of mercury bichloride overcomes all obstacles.

Much alluvial gold is found to be coated with a very thin film of iron oxide (rust), doubtless derived from the decomposition of iron pyrites.

This is most injuriously effective in preventing amalgamation, until removed by abrasion, and as very little abrasion is produced in the apparatus for catching alluvial gold, much metal is lost from this cause. In vein-mining no difficulty seems yet to have arisen in connection with coating by iron oxide.

But there is no doubt that the surfaces of gold grains have a remarkable affinity for sulphur, whether in the form of native sulphur or as sulphuretted hydrogen. Practically speaking, all vein gold is associated with more or less pyrites, so that a source of sulphuretted hydrogen is always present. There is good reason for believing that, in some cases at any rate, a chemical combination has taken place between the gold and the sulphur, so that the precious metal is no longer free, but in the form of a sulphide. In all cases of sulphurisation, the affection prevents such contact as is necessary to induce amalgamation with mercury, at least at ordinary temperatures, and even when in prolonged contact with boiling water. Heating to a temperature equal to the red heat of iron, whereby the sulphur is burned off, renders the gold particles amalgamable, either while still hot or after cooling. But if the gold is alloyed with much copper, an oxide of the latter metal will be formed by the heating in contact with air, and will spread as an invisible coating over the gold grains. The best remedy is to apply a solution of potassium cyanide, lime chloride, or free chromic acid.

When the gold occurs as an alloy with other metals which are not readily amalgamable, its affinity for mercury is so much reduced that it cannot be economically dealt with in the usual manner. The treatment thus rendered necessary is described in the next chapter.

(c) *Dirty Mercury*.—Mercury is only active whilst its surface is bright and clean, therefore anything which tends to sully it is an enemy to amalgamation. The so-called "flouring" or "sickening" of mercury is caused by antimony, arsenic, sulphur, arsenides, sulphides, sulphuretted hydrogen, sulphate of iron (green vitriol), and sulphate of copper (blue vitriol). The arsenides and sulphides probably do no harm till they undergo decomposition, which takes place slowly in the battery. The sulphates of iron and copper on some goldfields are very prevalent and injurious, apparently forming sub-sulphates with the mercury and adhering to its surface. One remedy suggested is to wash the vein-stuff before stamping, to dissolve out the sulphates present, but this means expense, and could not be really effective while the ore remained in large pieces.

Sludge Disposal.—The Board appointed in Victoria to deal with the sludge difficulty—which they rightly describe as "a difficult and intricate problem"—state in that portion of their Report relating to quartz mining:—"We do not, of course, desire to suggest anything

which would seriously hamper the mining industry ; but it is evident that the present mode of disposing of the detritus from quartz mines cannot, in fairness or justice, be allowed to continue. Having carefully considered the subject, the means we propose of preventing or mitigating future damage arising from this class of mining are the stacking or impounding of the tailings on the ground, the cost of which, the evidence shows, is not prohibitory ; the formation of dams to allow the tailings water from the batteries to settle ; and the abolition of all sluice-valves at the bottom of the dams, thus allowing the water to flow off from the top only. These precautions, if generally adopted, will prevent much of the mischief caused by quartz mining operations, we have not the slightest doubt."

CHAPTER VII.

TREATING REBELLIOUS AURIFEROUS ORES FOR RECOVERY OF THEIR
GOLD AND OTHER VALUABLE CONTENTS.

THE term "auriferous ores" is chosen in preference to the more common one of "pyrites" or "sulphurets," for while pyrites properly includes only the sulphides of iron and of copper, there are many other metallic ores which carry gold as a secondary constituent. In a previous chapter (pp. 101-107), these ores have been catalogued in some detail, but it will be well to repeat here that they comprise the sulphides, oxides, arsenides, and carbonates of the following principal metals: antimony, arsenic, bismuth, cadmium, chromium, cobalt, copper, iron, lead, manganese, tellurium, and zinc. Native silver is also always present in alloy with the gold. It may be said that there is not a single gold-mine in the world but what produces one or more of the metallic compounds alluded to, in addition to its free gold, and consequently that the extraction of the gold from such ores is of universal importance: indeed many mines depend almost entirely upon "pyrites," and yield scarcely any free gold at all. This problem of the economical separation of gold from the ores of base metals is surrounded with difficulty, and has been the subject of much study and experiment. The results of these investigations it is proposed to discuss in the following pages; but first it will be necessary to ascertain the composition of these so-called "rebellious" ores, and to examine the nature of the association between them and the gold.

Composition of Ores.

1. PORT PHILLIP (VICTORIA) PYRITES (Johnson and Matthey).

Gold	..	..	..	..	..	..	..	..	035
Silver	..	..	..	..	..	..	..	..	001
Arsenic	..	..	..	..	..	..	..	..	6850
Sulphur	..	..	..	..	..	..	..	..	6460
Iron, part in combination with the arsenic, sulphur, and oxygen, but chiefly in the metallic state	..	..	..	..	..	..	..	..	61250
Silica	..	..	..	..	..	..	..	..	15500
Oxygen	..	..	..	..	..	..	..	..	9000
Water of combination, and loss	..	..	..	..	..	..	..	..	904

100'000

The gold = 11'425 oz., and the silver = 0'325 oz., per ton of pyrites.

2. CLUNES (VICTORIA) PYRITES (J. Cosmo Newbery).

								1st.	2nd.
Silica ..	..	..	..	..	..	..	..	27'60	27'10
Oxide of iron (Fe_2O_3)	..	..	..	..	..	..	..	59'90	61'40
Sulphur ..	..	..	..	..	..	..	..	13'57	14'48
Lime.,	..	..	..	..	..	..	..	2'10	{not esti- mated.
Carbonic acid	..	..	..	..	..	..	..	not estimated.	

The mean results are 24'43 per cent. of iron pyrites (FeS_2), and 44'35 per cent. of oxide of iron; this latter is present as ferric oxide (Fe_2O_3), magnetic oxide (Fe_3O_4), and as carbonate (Fe_2CO_3), with some sulphate. The gold = 4 oz. 19 dwt. 23 gr. per ton of raw pyrites.

3. MARINER'S REEF, GYMPIE (QUEENSLAND) PYRITES (Dixon).

Copper ..	..	..	..	..	..	..	..	17'02
Lead ..	..	..	..	..	..	..	..	2'01
Antimony ..	..	..	..	..	..	..	..	3'90
Gold and silver ..	..	..	..	..	..	..	..	0'22
Iron ..	..	..	..	..	..	..	..	31'41
Sulphur ..	..	..	..	..	..	..	..	37'86
Silica ..	..	..	..	..	..	..	..	7'16
Arsenic and loss	..	..	..	..	..	..	..	0'42
								100'00

The gold = 12 oz. 10 dwt., and the silver = 62 oz. 9 dwt. 16 gr., per ton of pyrites.

4. GRASS VALLEY (CALIFORNIA) PYRITES (J. A. Phillips).

Sulphur ..	..	..	..	..	..	..	..	46'700
Arsenic ..	..	..	..	..	..	..	..	0'310
Iron ..	..	..	..	..	..	..	..	41'650
Copper ..	..	..	..	..	..	..	..	trace
Gold ..	..	..	..	..	..	..	..	0'037
Silver ..	..	..	..	..	..	..	..	0'036
Silica ..	..	..	..	..	..	..	..	10'970
								99'703

The gold = 12 oz. 2 dwt., the silver = 11 oz. 16 dwt., per ton of pyrites.

5. SONORA (CALIFORNIA) PYRITES (J. A. Phillips).

Sulphur ..	..	..	..	..	..	..	..	37'250
Arsenic ..	..	..	..	..	..	..	..	8'490
Iron ..	..	..	..	..	..	..	..	36'540
Lead ..	..	..	..	..	..	..	..	0'400
Gold ..	..	..	..	..	..	..	..	0'302
Silica ..	..	..	..	..	..	..	..	17'180
								100'162

The gold = 98 oz. 13 dwt. per ton of pyrites.

6. NORTH STAR (CALIFORNIA) PYRITES (J. A. Phillips).

Sulphur	43'720
Arsenic	1'360
Iron	39'250
Copper	0'220
Gold	0'026
Silver	0'012
Cobalt	0'150
Silica	14'230
	<u>98'968</u>

The gold = 8 oz. 10 dwt., and the silver = 3 oz. 18 dwt. per ton of pyrites.

7. COMSTOCK (NEVADA) ORES.

The Comstock ore is composed of native gold, native silver, silver-glance, stephanite, polybasite, rich galena, occasional pyrrargyrite, horn silver, and, with extreme rarity, sternbergite. Intimately associated with these, occur iron- and copper-pyrites and zinc-blende. Of these, pyrrargyrite and horn silver are rarities; polybasite and sternbergite, in recognisable crystals, occupy a few scattered localities; stephanite, in defined crystallisations, has been found in nearly every bonanza; but the main body of the ore is a confused semi-crystallised association of native gold and silver, vitreous silver ore, rich galena, copper- and iron-pyrites, and zinc-blende." The following two analyses are of samples from the Savage and Kentuck lower workings of 1869:—

ANALYSES OF COMSTOCK ORES.

	Savage.	Kentuck.
Silica	83'95	91'49
Protoxide of iron	1'95	0'83
Alumina	1'25	1'13
Protoxide of manganese	0'64	..
Magnesia	2'82	1'37
Lime	0'85	1'42
Sulphide of zinc	1'75	0'13
Sulphide of copper	0'30	0'41
Sulphide of lead	0'36	0'02
Sulphide of silver	1'08	0'12
Gold	0'02	0'0017
Bisulphide of iron	1'80	0'92
Potassa and soda	1'28	1'05
Water	2'33	0'59
	100'38	99'48
	W. G. Mixer.	A. Hague.

8. VICTORIA ANTIMONIAL ORES.

Antimonial sulphide .. 33 to 65 per cent. | Gold .. 1 oz. 10 dwt to 3 oz. 18 dwt. per ton.

9. MALDONITE.

The Costerfield antimony veins give considerable quantities of gold. The ore consists of sulphuret scamed with white and brown oxide of antimony, combined with a small proportion of auriferous quartz. The gold and antimony yields of some samples are thus stated :—

Locality.	Variety.	Antimony.	Gold.
		per cent.	oz. dwt. gr.
Ringwood	Sulphide	33	2 5 17
Costerfield	"	45	2 15 0
"	"	42	1 19 0
Sandhurst	"	63	1 10 0
Whroo	"	65	3 18 0
Newcastle, N.S.W. . .	"	"	0 15 0
Costerfield	Oxide	36	1 10 16

The Costerfield sulphide in smelting gave antimony containing silver, lead, copper, bismuth, arsenic, cadmium, manganese, cobalt, nickel, chromium, and iron.

10. RIO TINTO (SPAIN) ROASTED PYRITES (Gibbs).

Copper	..	..	..	..	..	..	..	1'65
Iron	..	..	..	..	..	..	..	3'64
Sulphur	..	..	..	..	..	..	..	3'53
Cupric oxide	..	..	..	..	..	..	..	2'75
Zinc	"	..	..	..	..	..	..	2'02
Lead	"	..	..	..	..	..	..	0'47
Silver	..	..	..	..	..	..	..	0'0037
Cobaltic oxide	..	..	..	..	..	..	..	0'007
Bismuth	"	..	..	..	..	..	..	0'013
Calcium	"	..	..	..	..	..	..	0'20
Ferric	"	..	..	..	..	..	..	77'40
Sulphuric acid	..	..	..	..	..	..	..	6'10
Arsenic	"	..	..	..	..	..	..	0'24
Insoluble residue	..	..	..	..	..	..	..	1'45
								<u>99'47</u>

* Calculated as Cu_2S and Fe_2S_3 . Gold in quantity too small to estimate.

11. THARSIS (SPAIN) ROASTED PYRITES (Gibbs).

Copper	..	..	..	..	..	..	..	1'50
Iron	..	..	..	..	..	..	..	3'23
Sulphur	..	..	..	..	..	..	..	3'15
Cupric oxide	..	..	..	..	..	..	..	2'56
Zinc	"	..	..	..	..	..	..	0'55
Lead	"	..	..	..	..	..	..	0'70
Silver	..	..	..	..	..	..	..	0'0023
Cobaltic oxide	..	..	..	..	..	..	..	0'032
Bismuth	"	..	..	..	..	..	..	0'010
Calcium	"	..	..	..	..	..	..	0'25
Ferric	"	..	..	..	..	..	..	77'00
Sulphuric acid	..	..	..	..	..	..	..	5'25
Arsenic	"	..	..	..	..	..	..	0'17
Insoluble residue	..	..	..	..	..	..	..	5'85
								<u>100'25</u>

* Calculated as Cu_2S and Fe_2S_3 .

12. SAN DOMINGOS (PORTUGAL) ROASTED PYRITES (Gibbs).

* {	Copper	..	..	..	..	..	..	..	1'55
	Iron	..	..	..	..	..	..	..	3'76
	Sulphur	..	..	..	..	..	..	..	3'62
	Cupric oxide	..	..	..	..	..	..	..	2'70
	Zinc	..	..	..	..	..	..	..	0'47
	Lead	..	..	..	..	..	..	..	0'84
	Silver	..	..	..	..	..	..	..	0'0023
	Cobaltic oxide	..	..	..	..	..	..	..	0'033
	Bismuth	..	..	..	..	..	..	..	0'013
	Calcium	..	..	..	..	..	..	..	0'28
	Ferric	..	..	..	..	..	..	..	78'15
	Sulphuric acid	..	..	..	..	..	..	..	5'80
	Arsenic	..	..	..	..	..	..	..	0'25
	Insoluble residue	..	..	..	..	..	..	..	1'85
									<u>99'31</u>

* Calculated as Cu_2S and Fe_2S_3 .

13. VITFERNEN (NORWAY) ROASTED PYRITES (Gibbs).

* {	Copper	..	..	..	..	..	..	..	1'01
	Iron	..	..	..	..	..	..	..	3'33
	Sulphur	..	..	..	..	..	..	..	3'10
	Cupric oxide	..	..	..	..	..	..	..	0'39
	Zinc	..	..	..	..	..	..	..	6'46
	Lead	..	..	..	..	..	..	..	0'06
	Calcium	..	..	..	..	..	..	..	2'30
	Ferric	..	..	..	..	..	..	..	68'06
	Sulphuric acid	..	..	..	..	..	..	..	6'56
	Arsenic	..	..	..	..	..	..	..	0'05
	Insoluble residue	..	..	..	..	..	..	..	8'74
									<u>100'06</u>

* Calculated as Cu_2S and Fe_2S_3 .

Condition of the Gold.—It seems to be generally agreed that the gold present in pyrites exists as metallic gold, though Prof. E. J. Chapman, of Toronto, asserts that the gold is present as an arsenide in the mispickel of North Hastings. At the same time, microscopic examination shows the gold to be in an extremely finely-divided state; and the evidence of Cosmo Newbery and Skey is to the effect that much of it is coated with a film of pyrites, so thin as to make no appreciable difference to the colour and lustre of the metal, yet sufficient to prevent its contact with the mercury, even after grinding. The want of uniformity in the opinions of the best authorities would seem to indicate that there is something yet to learn on this question of the nature of the association of the gold. In some ores the gold has been said to occur as a sulphide, and in others as a telluride.

TREATMENT OF COMPLEX ORES.

As the characters of the ingredients composing an ore must necessarily govern the processes suitable for its treatment, it will be convenient to discuss the subject in detail under the heads of the chief components.

ANTIMONIAL ORES.—The occurrence of gold in antimonial ores has been described on p. 576. From any of these it is possible to extract both the gold and the antimony. According to Cosmo Newbery, the following conditions are observed.

If antimonial ores are burnt in kilns or roasted in furnaces, either for the purpose of rendering the quartz more friable, or for getting rid of the antimony minerals, there is always a partial reduction, unless the heat is very great, and free access is given to atmospheric air. This reduction of ore producing metallic antimony is due to two causes—(1) the carbon of the fuel coming in contact with antimony oxide, either native oxide or that produced in the furnace by the oxidation of the sulphide, reducing it to a metallic condition; (2) by the action of the oxide on the sulphide, producing sulphurous acid and metallic antimony.

Metallic antimony has a great affinity for gold. It forms an alloy either when the two metals are melted together, or when the vapour of antimony is passed over heated gold. The alloy produced is grey in colour and very brittle, and amalgamates with mercury only after long contact and continual grinding, or by heating the two together. The amalgam, when formed, floats on mercury, and gradually gives up metallic antimony as a fine powder when agitated with water. This antimonial powder carries off a quantity of mercury and gold-amalgam entangled with it.

Antimony sulphide is perhaps the worst mineral with which the quartz-crusher has to deal. It divides the mercury into a black "flour" even more quickly than arsenical pyrites; and if this flour is triturated with the intention of bringing the globules of mercury together, a chemical combination takes place. The mass gradually changes colour, passing from the original blue-black or dark grey to a pure black, and then through brown to a brown-red. Upon examination, Cosmo Newbery finds that the remaining mercury contains antimony, and that the brown-red non-metallic portion consists of a mixture of undecomposed antimony sulphide and mercury sulphide.

Sodium amalgam he finds to be worse than useless in bringing together the globules of mercury floured by antimony sulphide. When only containing a small percentage of sodium, it has no action; when

made stronger (sufficient sodium to cause a slight evolution of hydrogen when the mercury is placed in water), it decomposes, "the antimony sulphide forming sodium sulphide, an amalgam of antimony, mercury, and sulphuretted hydrogen." Sodium amalgam also reduces the metal from antimony oxide. If antimony sulphide is fused with finely divided gold, a portion of the gold enters into chemical combination with the ore, and is dissolved with the antimony sulphide in alkaline solutions. Antimony oxide has no effect on mercury or amalgam.

The process adopted by the Costerfield Co. for treating their ores, consisting of sulphide and brown and white oxides of antimony (see p. 576), is as follows. The portion of the ore free from quartz is picked out and set aside for smelting, the remainder being crushed to extract the gold. The tailings, which consist of antimony and a little quartz sand, are then conveyed to heaps, and are subsequently prepared for smelting by a process of buddling. A sluice-box, into which a stream of water is turned, is fed with tailings, which are made to pass thence on to a triangular tray forming an inclined plane, so arranged as to cause the water and tailings to flow over it in a broad shallow stream, into an oblong receiving-pit below; the purest antimony-ore, from its greater specific gravity, settles in the pit at the end nearest the tray; as the sediment recedes from this end, it gradually becomes mixed with an increasing proportion of sand, but much of the latter is carried away in the overflow of water from the pit.

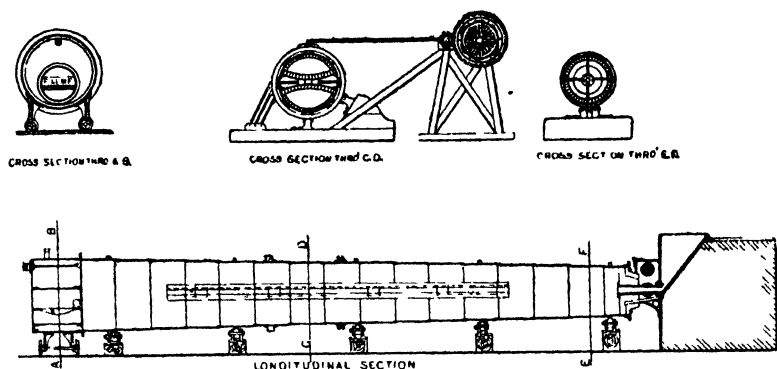
On cleaning out the receiving-pit, that portion of its contents containing quartz sand is returned to the heaps, to be again passed through the buddle, and the pure ore is collected in bags, and sent to the boiler-house to be dried; it is then placed in a smelting-furnace, with equal proportions of uncrushed ore, and reduced to crude antimony (sulphide), the slag and cinder resulting from this process being further treated by roasting or calcining in a reverberatory furnace to liberate the oxide, which passes off in fumes from both furnaces into the oxide flue; and as the fumes cool on their passage to the smoke-stack, the oxide is deposited in chambers constructed in the flue to receive it. The residuum from the reverberatory furnace is afterwards crushed to extract any gold it may contain.

The gold obtained from the mineral defrays the whole of the company's working expenses, and the yield of crude antimony and oxide is clear profit; none of the latter has yet been brought into the market, but the shipments of rough ore hitherto sent to England have realised 9*l.* to 12*l.* per ton. The ore yields about 45 per cent. of crude antimony, which it is expected will fetch 20*l.* to 22*l.* per ton in London.

The revolving furnaces used in the foregoing operations were devised

by H. Herrenschmidt and Borthwick, and treat 20 to 50 tons per diem. Herrenschmidt's is shown in Fig. 240. It measures 35 ft. long, 3 ft. in diameter at one end and 4 ft. at the other, and is made of $\frac{1}{2}$ -in. plate.

FIG. 240.



HERRENSCHMIDT'S REVOLVING FURNACE.

The process adopted for treating auriferous ores containing antimony sulphide, by fusing the sulphide with a portion of metallic antimony, and using the same metal with fresh charges of the ore until it becomes rich in gold, and then separating the two metals by the oxidation of the antimony, while suitable for rich antimony-ores, will not answer for those containing less than 30 per cent. of the sulphide, as they are too silicious to fuse. Hence very large quantities of the poorer auriferous antimony-ores do not yield half their gold to ordinary processes, and do not return any antimony when worked for gold.

Cosmo Newbery has introduced the following method for treating poor auriferous antimony ores, which may also contain gold, silver, nickel, cobalt, sulphur, and arsenic. The uncrushed ores are placed in a kiln or furnace, with a quantity of salt sufficient to produce the amount of chlorine necessary to get rid of the sulphur, antimony, and arsenic. As soon as the calcination commences, a supply of steam or aqueous vapour is conducted to the bottom of the kiln or into the furnace, in such quantities as to keep the whole mass saturated. That it is so saturated is ascertainable by holding a condensing surface, such as a piece of cold iron, over the calcining mass: if the saturation is being effected, the surface will become damp. The saturation is continued until there are neither antimonial nor arsenical fumes, nor the smell of sulphurous acid or sulphuretted hydrogen. The process is then completed, and the charge is drawn; it is ready for any further treatment for extracting the precious metal.

A condenser for facilitating the solidification of the metallic vapours given off in these roasting processes has been perfected by Cosmo Newbery and John Lister Morley.

To accomplish this object, it is essential that the fumes should be directed with considerable velocity against a series of solid resisting surfaces, and finally against a web of permeable material, through which the permanently gaseous portion may filter. The greater the force with which such fumes impinge against a series of metallic discs, the larger is the percentage of solids which attach themselves to such discs, and ultimately fall into a receptacle beneath. Also, if such current of fumes is concentrated so as to be compelled to pass through a narrow opening, and then against a metallic disc, the effect is heightened. If the velocity of the current be great enough, that is to say if the draught be strong enough, by far the larger proportion of the solids will deposit themselves on striking the discs, leaving only a small portion to be retained by the final filter; whilst on the other hand, if the discs were removed, the whole of the solids would be retained by the final filter, but would give it unnecessary work to do.

This filter is made of metallic gauze, which may be protected from the action of the fumes by a coating of any non-volatile oxide infusible at the temperature which it is required to withstand. The mesh of this gauze is reduced to any requisite fineness by felting with asbestos or covering with asbestos cloth, or, when the fumes are sufficiently cool, with wool or vegetable fibre.

An illustration of how an apparatus might be constructed for the purpose of carrying out this method is shown in Fig. 241. In the flues *a* are a series of resisting surfaces (discs) *b*, *c* being the orifices through which the gaseous current would flow and impinge on such surfaces, and *d* the final filter; *e* are the receptacles for the solids which fall after impinging against the discs, and which can be removed through a door *f* at the bottom. The necessary draught is produced by artificial means, although it does not matter how it is produced so long as it is strong enough.

The apparatus is applicable to the condensation of fumes of antimony, arsenic, bismuth, lead, and zinc. The process is simple inexpensive, and effective. The solid particles adhering to the plates, the arsenic, antimony, and other volatile oxides, form a dense cone of such solidity that the strongest current of air fails to dislodge any. Antimony oxide collected

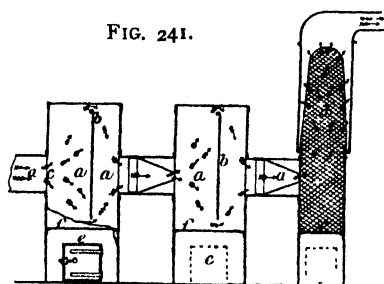


FIG. 241.
FUME CONDENSER.

in this manner is well suited for the manufacture of paint, if the temperature of the fume has been lowered from the heat of the furnace in which it is generated till the vapour has been all condensed into solid particles of oxide in suspension. If the vapour of the oxide condenses on the plate, it is no longer suited for paint, being crystalline, and having no "covering" power when mixed or ground with oil. The arsenic condensed by this method should also be a marketable commodity, and, as crude arsenic, become of value to the pyrites worker, instead of a public nuisance.

Designolle's Process.—Designolle's process is based upon the following grounds: (1) that when an acid solution of any salt of mercury is exposed to the action of an electric current, the salt will be decomposed, and metallic mercury will be deposited at the negative pole, amalgamation resulting when that pole is formed of gold; (2) when scraps of clean iron are in contact with a slightly acid solution of a salt of mercury, on touching these with gold, the salt of mercury is decomposed, metallic mercury combining with the gold; (3) the great affinity of chlorine for antimony and tellurium. Reasoning from these conditions, the process consists in using a bichloride of mercury solution in water containing sodium chloride, whereupon antimonial gold and the telluride of gold are decomposed, the chlorine uniting with the antimony and tellurium, while the liberated metallic mercury amalgamates with the gold.

ARSENICAL ORES.—Arsenic occurs largely in auriferous ores in combination with iron and sulphur (as arsenical pyrites, or mispickel), also as arseniates of lead, copper, and iron. From any of these minerals it may be easily set free in the roasting-furnaces and quartz-kilns, especially the latter, owing to the fuel, and the reducing gases produced by the imperfect combustion, coming into contact with these ores. When the arsenic is thus set free, it passes through the furnace as vapour, and is readily taken up by the gold that may be present; for when gold is heated to redness, or any degree between that and its melting-point, it takes up arsenic, with which it forms a grey, easily-fusible, brittle alloy; even when the arsenic is not in sufficient quantity to change the colour or fusing-point of the gold, it still renders it very brittle; one thousandth part renders gold so brittle that it may be ground to a powder. Gold containing arsenic is more difficult to amalgamate than pure gold. If much arsenic is present, the amalgam is powdery and black, and floats on the surface of the mercury; the black colour is due to the separation of arsenic. This black powdery metallic arsenic does not unite at ordinary temperatures with mercury to form an amalgam, but it mixes with it, coating every globule with the black powder, thus preventing their uniting with each other, or, in other words, causing the mercury "to flour." Sodium amalgam aids the union of mercury floured by metallic

arsenic; but if arsenious acid (common white arsenic) is present, it reduces it to the state of metallic arsenic.

Arsenical pyrites acts seemingly in the same way as metallic arsenic with mercury; when ground together, a large amount of black floured mercury is produced. If the pyrites is partly decomposed, this action is more energetic than with the original mineral. Cosmo Newbery can detect no actual combination with the mercury. The black coating, examined under the microscope, only seems to be a mixture of pyrites-grains and mercury-globules, both very finely divided. When mercury covered thinly with this black coating is warmed, the coating is absorbed into the mass, and is liberated again as the mercury cools.

The liberation of the arsenic from auriferous ores is most commonly effected by means of roasting them in furnaces of various construction, which will be now noticed, premising that Newbery's process described under Antimony (p. 580) is equally applicable.

Chapman's Process.—The following process was introduced by Prof. E. J. Chapman for treating the mispickel ores of Marmora, Ontario. These ores contain 7 to 8 dwt. of free gold per ton, and a total of nearly 7 oz. per ton, but for a long time their treatment could not be made remunerative.

Chapman discovered that by deflagrating the ore with nitrate of soda, a most rapid absorption of oxygen takes place, and a large proportion of the arsenic at once combines with the nitrate of soda, and becomes soluble in hot water. The sulphur being burnt is, to a great extent, driven off, and the iron by absorption of oxygen rapidly becomes magnetic. The leachings of soluble arsenic subsequently obtained contain large quantities of sulphate of soda, which, rapidly depositing in a crystalline form, admits of the soluble arsenic being drawn off, precisely in the scale required for the manufacture of Paris green, when combined with sulphate or other salts of copper in the ordinary manner. If the manufacture of orpiment is required, the arseniate of soda combined with sulphuretted hydrogen deposits a pigment of a beautiful yellow colour. In either case, the resulting by-product is worth the cost of the materials used in the manufacture.

The process is carried out as follows. The ore, after being very finely ground, is sifted through fine wire sieves, so as to prevent the presence of any large particles. It has been found that whereas it will require 12 hours' roasting to expel the arsenic from a cube of ore $\frac{1}{8}$ in. square, if the cube be divided into 10 particles, one-tenth of the time will suffice. The sifted ore is mixed with 10 per cent. of its weight of nitrate of soda dissolved in water. The ore is then fed slowly into a peculiarly shaped iron retort, constructed in 2 half-circles, with flanges at each side, to withstand any tendency to warp, and also to form a resting-place for

the retort when set in brickwork. The retort is 10 ft. long by 30 in. in diameter, cast in a peculiar manner, so as to avoid cracking and resist wear. It is set in brickwork, with an inclination towards its mouth of about 8 in. The fireplace is at the mouth, and the heat traverses the entire length of the retort, ascends at the rear, and passes forward and over the upper half, until it escapes into the chimney. The retort is thus enveloped in fire sufficiently powerful to keep the contents at a dull-red heat. The fuel required is not much, as the combustion of the sulphur and nitrate heats the ore to redness at once, and after it is hot, a very small fire will keep it so.

The ore, being fed into the rear end, gradually works its way downwards towards the mouth, each revolution of the internal machinery combining with the inclination to produce this effect. The machinery that generates this motion is a shaft that passes through the entire length of the retort, and is made to revolve half round by means of a crank attached to the projecting end ; strong arms are fastened to the shaft, and attached to them are iron bars, so arranged that, as the shaft revolves half round, these bars scrape the bottom of the retort, and carry the ore to and fro, first to one side and then to the other, thus causing it to flow over each bar during each revolution in a thin stream of red-hot ore, much the same in appearance as a broad fall of water. This motion causes all the disengaged arsenic to pass off very readily. Arsenic in this state practically sublimes at about 500°C. (932°F.) ; but it condenses quite easily, and unless means are provided for its free disengagement, is a long time before it passes quite away into the condenser provided to receive it.

Being fed into a red-hot retort, the ore instantly deflagrates, and the furnace is filled with blue flame ; some arsenic begins to pass over into the condenser, but a large proportion combines with the nitrate, and this portion becomes soluble in hot water, and can readily be removed by the filtration that takes place afterwards.

The speed of the shaft must be so arranged that the ore is delivered at the mouth of the retort in about $\frac{1}{2}$ hour. As the ore leaves the retort, it falls—whilst red-hot—into water. The effect of this is to boil the water, to dissolve out the arsenic made soluble by the nitrate, as well as to burst the particles of ore and reduce them still further to a smooth paste, and thus render the gold more attainable. The liquor that results from the leaching rapidly deposits—in a crystalline form—quantities of sulphate of soda. The remaining liquid is now fit to yield Paris green, or orpiment, as required. After leaching, the ore is returned to the roasting-retort ; this is very similarly constructed to that used for deflagrating, but, being set at a less inclination, the ore does not reach the mouth of the retort until it has had at least 2 hours' more roasting. The arsenic is now

altogether driven off, the ore assumes a deep-red colour, and the iron having become magnetic and parted with all the arsenic, the combination no longer influences the amalgamation of the mercury and gold.

Cule finds that the old German barrels give the best results as amalgamators. Their action is greatly assisted by the use of steam, which Cule has applied in pipes that pass and repass through the barrel, and by the heat keep the ore always at about 212° F. (100° C.), consequently the mercury is kept quite limpid, and in that state will more readily catch the millions of minute particles of gold that pervade the prepared ore, and which have been loosened, so to speak, from amongst the other constituents of the mispickel, by the oxidation of the ore, and destruction of the combination of arsenic, sulphur, and iron that contained it. Cule also uses stones, and finds their action most beneficial in reducing the prepared ore to a still finer soft paste, and also in causing a certain abrasion or scouring of the surface of the particles of gold, rendered somewhat sullied by the roasting and nitrate, in which state the mercury does not so readily attack it.

Cule finds the result of all this to be : that mispickel ores that contain about 50 dwt. of gold to the ton can be readily made to yield at least 40 dwt. ; that the bye-products—soluble arsenic, arsenious acid, sulphate of soda, and oxides of iron—are believed to be fully as valuable as the cost of manipulation of the ore, including mining, thus leaving the gold as almost all profit. As a further bye-product, the sulphurous acid fumes can be caught and condensed in chambers, and will be useful for the manufacture of superphosphate of lime. The nitric acid also will be forced into contact with lime, and a valuable manure will probably result from this course.

While admitting the probable increase of gold return by this process as compared with those in general use, Cosmo Newbery doubts its applicability in Australia, for the following reasons. Assuming the pyrites to contain 2 oz. of gold per ton, of which 80 per cent. (the Port Phillip Co. extract 95 per cent.) is recovered, this would mean a loss of 8 dwt. of gold per ton, worth (at 4*l.* per oz.) 32*s.* The amount of nitrate required is 2 cwt. per ton, costing 24*s.* a cwt. (infinitely more since the border difficulty in Peru and Chili), so that, excluding labour and fuel, the 32*s.* worth of gold would need 48*s.* worth of nitrate of soda, even supposing all the present loss to be saved. This loss of 16*s.* would have to be recouped from the sale of arsenical pigments, which have as yet no value in Australia.

Flude's Furnace.—The author is indebted to Joseph Flude, Esq., Superintendent of Laboratories at the School of Mines, Ballarat, Victoria, for the following observations, and description of Serjeant and Flude's furnace :

"On all our gold-fields there is a considerable quantity of pyrites associated with the quartz, in some cases existing as sulphides, in others as arseno-sulphides. In order to extract the gold (which is always associated with the pyrites) from them, various methods have from time to time been devised; but at present the most popular, and, I think, the best, process is that of roasting to sweetness. This operation is performed by submitting them to a dull-red heat for a lengthened period under constant rabbling in long furnaces, the hearth of which is inclined at about 1 in. per foot; the charge enters at the upper end and the fireplace is at the lower end of the furnace; the rabbling causes the charge to gradually descend the inclined hearth, so that when it approaches the bridge of the furnace it receives the greatest heat, and is finally discharged (by intermittently withdrawing a slide) through the hearth into a chamber under the bed of the furnace. By this method, when carefully conducted, the pyrites can be roasted to almost perfect sweetness, so that by a corresponding care in the after-manipulation, nearly the whole of the gold may be extracted.

"It may readily be seen that in such a furnace the sulphur and arsenic in the pyrites take fire, are converted into sulphur dioxide and arsenious oxide at the expense of atmospheric oxygen, pass away together with carbon dioxide from the fire, and escape at the top of the chimney-stalk into the air. The arsenic is more or less effectually condensed in a long horizontal flue, between the furnace and chimney-stalk; these flues are opened at stated periods and cleaned out, and the arsenious oxide is deposited in any convenient place, so that it may be got rid of.

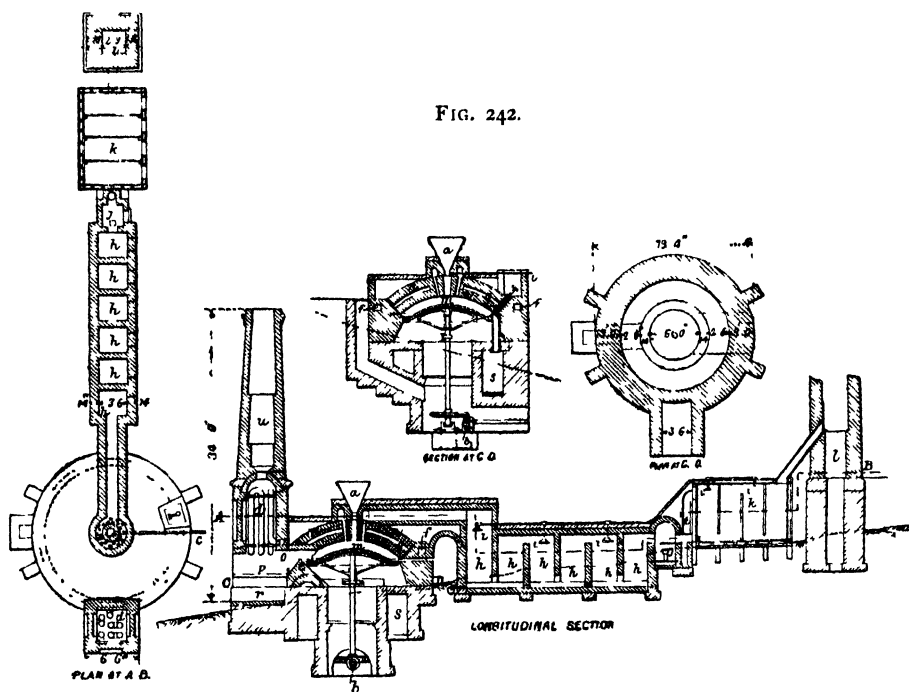
"There are at least two objections to this plan of operations:—(1) all the sulphur and arsenic are lost; (2) it is injurious to health and vegetation, by polluting the surrounding atmosphere and streams of water.

"The furnace to be described (Fig. 242) is a modification of the Brunton, and is intended to meet all objections to the furnaces as at present in use in Australia. It is almost automatic, being self-acting in so far as the feed and discharge are concerned, and, when set at work at the proper speed, is continuous in its operation; the only care required is to keep the hopper *a* supplied with the material to be roasted, consisting of pyrites which has passed through the stamper-box grating of say 100 apertures to the sq. in., and afterwards concentrated to nearly clean pyrites.

"The bed of the furnace is worked by belt and pulley on a shaft connected with the bevel-pinion *b*, and driven by the machinery which grinds and amalgamates the roasted pyrites. On the same shaft is another belt and pulley, which is intended to drive the self-acting feed-shaft *c*, the end of which passes through the throat of the hopper, so

arranged that at every revolution a quantity of the pyrites is discharged into the furnace, sufficient to keep it supplied with fresh material, which, by the time it arrives at the periphery of the hearth, is completely roasted; thence it is discharged down fluming into a chamber below, where it is allowed to cool, and is passed into the tritulating-machine.

“The novelty of this furnace consists in the fact that no fire passes over the materials when in course of roasting, but, instead, an abundant



SERJEANT AND FLUDE'S FURNACE.

supply of heated air is used, which passes through heated pipes in a chamber *d* immediately over the fire-place, and cut off from the bed of the furnace by closing a strong damper in front of the fire-bridge *e*. The heated air from the pipes enters a flue *f*, which continues round the entire crown of the furnace, communicates at intervals with the chamber *g* in the crown, and leaves this chamber through perforations in the bricks, causing the air to impinge on the hearth: thus securing the thorough oxidation of the arsenic as arsenious oxide (As_2O_3), of the sulphur as sulphur dioxide (SO_2), and of the iron as ferric oxide (Fe_2O_3).

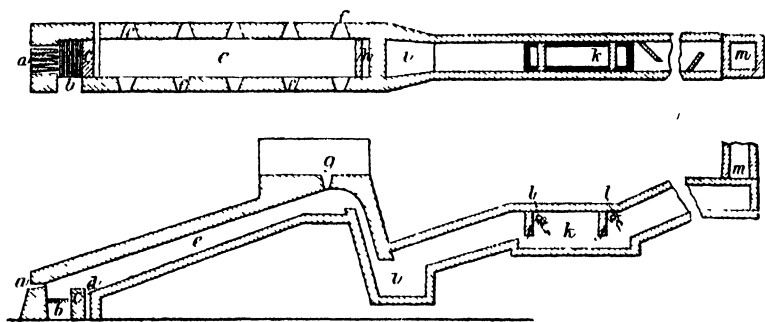
“The SO_2 and vapour of As_2O_3 pass into the chambers *h*, and the arsenic as arsenious oxide (As_2O_3) is condensed by means of steam-

jets *i*, while the sulphur dioxide continues through the furnace *j*, where nitrous gas is generated; the mixed gases of sulphur dioxide, nitrous acid, and atmospheric air pass on into the sulphuric acid chamber *k*, where they finally become converted into sulphuric acid, the waste gases being allowed to escape up the chimney-stalk *l*.

"The various parts are marked thus:—*a*, feed-hopper; *b*, bevel-gearing to drive furnace-bed and feeding apparatus; *c*, shaft of self-feeder; *d*, hot-air pipes over fireplace; *e*, fire-bridge; *f*, hot-air flue communicating with chamber *g*; *g*, chamber in crown of furnace; *h*, condensing chambers for arsenic; *i*, steam-jets; *j*, small oven for generating nitrous gas; *k*, leaden sulphuric-acid chamber; *l*, chimney-stalk, 62 ft. high; *m*, furnace hearth; *n*, furnace crown; *o*, damper; *p*, fireplace; *r*, ash-pit; *s*, chamber to receive roasted pyrites; *t*, flue for passage of mixed gases into sulphuric-acid chamber; *u*, chimney-stalk of furnace; *v*, discharge."

Port Phillip Co.'s Furnace.—The reverberatory furnace used by the Port Phillip Co. for treating their arsenical pyrites is of the simplest description, as illustrated in Fig. 243. *a* are air-tubes for admitting air

FIG. 243.



PORT PHILLIP CO.'S FURNACE.

above the fire; *b*, fireplace; *c*, fire-bridge; *d*, discharge slot for burnt pyrites; *e*, roasting hearth; *f*, holes for insertion of a spade-bar to turn the pyrites over; *g*, hopper for charging pyrites into the furnace; *h*, flue descending to the pocket *i*, where heavy condensable matters deposit; *j*, stops for checking the flow of gases; *k*, leaden cistern wherein the arsenious vapours can condense, the operation being aided by the spray-pipes *l*; *m*, chimney for final escape of uncondensed vapours.

This furnace is perhaps better known as Latta and Thompson's, from the names of its inventors. It requires only one attendant per shift, and the labour may be reduced by making the furnace in 3 sections at right

angles to each other, so as to facilitate the operation of working the charge through. The roasting must be continued till all the sulphur and arsenic are driven off, and the pyrites becomes "sweet"; but the heat must not be carried too high, nor is it necessary to convert all the iron present into sesquioxide. The time occupied in the process is 12 to 18 hours, but efficiency must not be sacrificed to time, for partially decomposed pyrites causes more flouing of the mercury, and greater loss of gold and amalgam in the subsequent stages, than when raw. The sweetness is judged by the roasted material emitting neither odour nor fumes, by its giving off no sparks while hot, by its turning red when cold, and by its ceasing to run easily when stirred. About 4 to 4½ tons of pyrites per diem can be treated in such a furnace. The consumption of fuel will depend much upon the regularity of the stoking. Great attention must be paid to the draught of the furnace, as it is essential that a steady current of air should pass over the surface of the pyrites, to ensure thorough oxidation. Opinions are divided as to whether the presence of a small percentage of sand with the pyrites is detrimental or beneficial: on the one side, it is maintained that the sand tends to check the agglutination or caking of the particles of pyrites, which, when once begun, makes it very difficult, if not impossible, to roast properly; on the other hand, it is contended that the sand cuts up the mercury, and causes a great loss in the subsequent process of amalgamation, especially if a Chilian mill is used. Mixing charcoal with the pyrites in roasting promotes its decomposition by the combination of the sulphur with the carbon; but a good furnace properly attended requires no such aid, and when lead or antimony is present in the pyrites, the carbon acts injuriously by reducing the metals from their ores.

The second step in the treatment is to grind and amalgamate the calcined ore in the pan of a large Chilian mill (see p. 494). Mercury is added for this purpose, and the grinding helps to break up the mercury and bring it into contact with the liberated gold particles.

The third stage is the collection of the particles of gold, mercury, and amalgam escaping from the mill. This is effected by passing the whole through a concentrator, or even two in succession, and then through a mercury-trough and amalgamating-barrel, the tail-sand being run through buddles. By these means, as high as 95 to 98½ per cent. of the gold in the pyrites has been extracted.

The following results were obtained with the pyrites treated at the Port Phillip works for the year ending Oct. 10, 1877:—

Pyrites, raw	..	..	..	..	..	..	321 tons 10 cwt.
Gold produced	..	..	..	..	..	..	1549 oz.
Average per ton (raw)	..	..	..	..	..	..	34 oz. 16 dwt. 9 gr.

Loss in weight by roasting, say 20 per cent.

Costs per ton—							£	s.	d.
Buddle expenses	..	..	..	..	..	..	1	1	8
Roasting—Labour, 13s. 3d.; fuel, 9s.	..	..	..	..	..	..	1	2	3
Grinding, labour	..	..	..	..	..	..	0	13	3
Mercury lost, 1 lb. 6½ oz., value	..	..	..	..	..	..	0	4	3
Total cost per ton							3	1	5
Cost per oz. of gold obtained							0	13	2
Mercury and gold recovered in the steam barrel for 8 months in use—									
Mercury, 185 lb.; gold, 55 oz. 5 dwt.									

At the Walhalla Co.'s works, the same treatment with slight alterations was introduced in July 1868 and carried on until February 1876. During that time, 1411 tons of raw pyrites, yielding 942 tons of calcined ore, were treated, which returned 2476 oz. 11 dwt. 6 gr. of standard gold, or at the rate of 2 oz. 12 dwt. 14 gr. per ton of calcined ore; the loss of mercury being 1839½ lb., that is 1·92 lb. per ton of calcined ore.

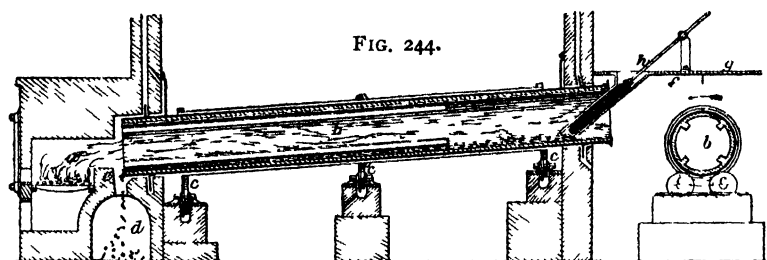
The cost of roasting the ore and other incidental expenses connected therewith was—

	s.	d.
Firewood	13	0
Wages	16	7
Repairs, cleaning flue, &c.	4	2
	33	9

It may be incidentally mentioned that while the average standard of the vein-stone gold of Walhalla is 20·1 $\frac{5}{8}$ to 20·1 $\frac{7}{8}$ carats, the gold from unroasted pyrites is only 19·3 $\frac{7}{8}$ to 20·0 $\frac{3}{8}$ carats, whereas that from roasted pyrites is 22·0 $\frac{4}{8}$ to 22·1 $\frac{1}{8}$ carats.

Revolving Furnaces.—The inclined reverberatory furnace requires a considerable expenditure of labour for passing the charge through, hence several forms of mechanically revolving furnace have been introduced. Hocking and Oxland's may be described as typical of this kind of furnace. It is shown in Fig. 244, and consists of a fire-box *a*, whence the heat and products of combustion pass through an iron tube *b*, made of boiler-plate and lined with fire-bricks on edge. The tube, which is 30 to 40 ft. long, is supported in an inclined position, but varying in inclination according to the character of the ore treated. It is supported on 3 pairs of friction-wheels *c*, and rotated by gearing in the middle. It enters the fire-chamber, and is so arranged as to deliver the ore passing through it by an opening *e* into the chamber *d*. At the upper end it communicates with flues or condensing chambers *f*, and the ore is dried on cast-iron plates *g* covering these chambers, being fed into the hopper *h* by a boy who attends to the fire. The tube revolves at a rate of 3 to 8 revolutions per minute. The ore is raised by 4 projecting lines of bricks parallel with the axis, leaving room for the continuous running-in of the dry ore from the hopper. When the ore has been raised sufficiently high on one

of these shelves, it falls off in thin streams through the hot gases passing up the tube. It thus becomes sufficiently heated for the sulphur and arsenic to take fire, and to burn with such energy that before the ore arrives halfway down the cylinder, the greater portion of the arsenic and much of the sulphur is driven off. The heat evolved by the combustion of arsenic and sulphur is thus rendered available for heating the upper portion of the tube. Throughout the whole of the tube, the lines of shelf



HOCKING & OXLAND'S REVERBERATORY FURNACE.

perform the duty of passing the ore in finely-divided streams through the heated gases in such a way that no particle can escape full exposure to the oxidising influences required for perfect calcination. It is found that arsenic burns first, and that its removal is completed some time before the last portions of sulphur are eliminated. The calcined ore, passing from the lower end of the tube into the burnt-ore chamber at a bright-red heat, contains only traces of arsenic, and but a small proportion of sulphur.

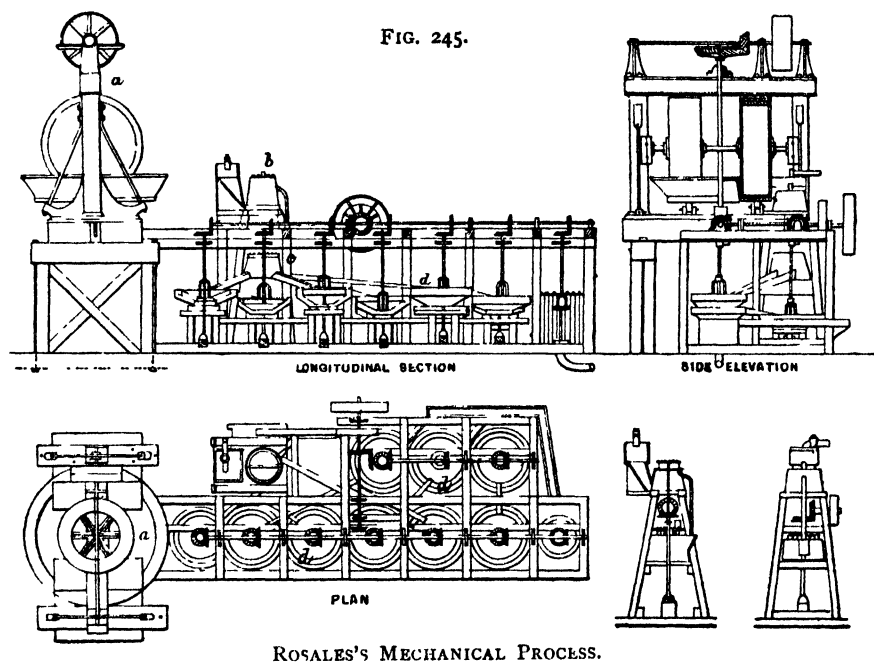
Rosales's Process.—In the belief that the expenditure for roasting the pyrites, as just described, was unnecessary for the liberation of the gold, Henry Rosales, the talented manager of the well-known Walhalla mine, experimented in the direction of treating the pyrites by fine mechanical subdivision without roasting. The outcome of his experiments was the successful plan now to be described from documents very obligingly furnished by him to the author.

The mode of operation without roasting is—(1) to grind the pyrites to a fine pulp, in a Chilian mill for instance, and (2) to amalgamate in separate vessels, such as dollies, in conjunction with improved Tyrolese or Hungarian mills. At the Walhalla Co.'s works, the process is worked in the following manner. (See Fig. 245.)

The concentrated pyrites, which has been exposed to the action of the atmospheric air* during 3 to 6 months, and consists mainly of iron-pyrites (FeS_2), arsenical pyrites, mispickel ($\text{FeS}_2 + \text{FeAs}_2$), and more or less

* This exposure might be supposed to cause a considerable oxidation of the pyrites, but Rosales assures the author that in this case it is not so, there being no appreciable change in the composition, though the pyrites suffers a physical change which allows it to be ground to a much finer and softer pulp

quartz-sand, is subjected to grinding in a Chilian mill *a* (the rollers of which are 18 in. wide, making $13\frac{1}{4}$ revolutions per minute) to a very fine state of disintegration. This is effected in the following manner. The charge, consisting of $3\frac{1}{4}$ buckets of dry pyrites, weighing 107 lb., is, after 7 pints of water (less if the stuff is a little moist) have been filled into the pan, gradually emptied into the pan whilst it is revolving. Should the stuff be too dry, and the rollers run on top of the pyrites for more than 2 minutes, $\frac{1}{4}$ to $\frac{1}{2}$ pint of water is slowly poured on until the rollers get on the bottom of the pan, which should be the case about 5 minutes after the charge has been put in. The main thing after this is to keep the stuff well together, forming a ridge in the middle of the pan by self-adjusting



scrapers placed on each side of the pan and of the rollers, and also scraping with a shovel those places which the scrapers have failed to clean. About $\frac{1}{4}$ hour after the charge has been working, 2 oz. of caustic soda are distributed in the pan. After one hour's grinding, when the stuff commences to get stiff, 3 to $3\frac{1}{4}$ half-pints of water are added at different times during the last $1\frac{1}{2}$ hour that the stuff is being ground. For the last 15 minutes, if the stuff has been properly attended to, its appearance is a fine pulp having a glossy and silky appearance. It is then ready to be shovelled out of the pan. But until its appearance is glossy and silky, the charge should not be taken out, as it would not be sufficiently ground.

The ground stuff is shovelled into a tub, where it is kept in a moist and pasty state, not too stiff, and whence it is passed on to a basin standing on a grating of 81 holes to the sq. in., through which it passes with a stream of water delivered into the basin, at the rate of 36 gal. per minute, and is conveyed into the top force dolly *b*, which is charged with 60 lb. of mercury, and revolves at the rate of $33\frac{1}{2}$ revolutions per minute. Each charge is passed into this dolly from the tub in $2\frac{1}{2}$ hours in regular proportions of about $\frac{1}{4}$ shovelful at a time, so that the stream of 36 gal. of water per minute should hold in suspension and carry with it about $\frac{3}{4}$ lb. of pyrites per minute.

From the top dolly, the stream flows into the bottom of the lower dolly *c*, which is also charged with 60 lb. of mercury, and revolves at the same speed as the upper one. The overflow, or the stream from the lower dolly, is divided into three sets of three Tyrolese mills (see p. 501) each *d*, which are placed one below the other. Each mill is lined on the working surfaces with amalgamated copper-plates, and is charged with mercury to the lower edge of the copper-plates, which requires 27 to 35 lb. of mercury. The runner revolves at the rate of $19\frac{3}{4}$ revolutions per minute, and is set so that the bottoms of the arrows are $\frac{1}{8}$ in.—i. e. $2\frac{1}{2}$ to 3 turns of screw on the spindle—above the mirror of the mercury. Thus the stream passing through each set of Tyrolese mills is equal to 12 gal. of water, carrying $\frac{1}{4}$ lb. of ground pyrites with it.

To ensure the proper working of the mercury in the dollies and mills, sodium amalgam is used at the rate of $2\frac{1}{2}$ lb. per 10 tons of dry pyrites treated: that is to say, 4 oz. per ton. A small quantity (about the size of a pea) is dropped every 8 hours into the mercury of each of the lower Tyrolese mills, and slightly more into that of the upper ones and of the dollies. The waste water from the third Tyrolese mill of each set flows into the creek. This company, unfortunately, has no convenience to erect settling-pits, &c., to catch the waste pyrites; but where convenient, and especially in a large establishment, it might be advantageous to save the waste pyrites for after-treatment, as owing to insufficient grinding, a certain percentage of comparatively coarse pyrites could thus be saved.

As described, charge after charge is ground and subsequently run through the amalgamating-mills during one working month—26 days. Every day, however, the light dross amalgam, which does not sink into but floats on the surface of the mercury of the two dollies, is removed, together with the heavier and not sufficiently-ground pyrites. The dross amalgam or alloy is washed out in an enamelled dish, and kept by itself until the end of the month, whilst the pyrites that has been removed is at the end of the month re-ground with the last charges. The quantity of pyrites thus daily collected is only one-third of an enamelled bucket, about 15 to 20 lb., if the stuff has been well ground. At the end

of every month, all the dollies and Tyrolese mills are emptied and washed, and the mercury of each dolly and mill is separately collected and passed through a chamois-leather bag, and the amalgam in each case is separately retorted. Whenever the ore had not been sufficiently ground—on account of the mill not grinding well, or because it was attempted to pass through too heavy a charge in a given time, or that too little water was passed, or that too much ore was rushed through the Tyrolese mills—a loss in mercury and gold was the immediate result. As a proof of the very satisfactory manner in which the improved Tyrolese mills worked, it may be pointed out that of 105 oz. 17 dwt. of gold, which passed through the second dolly, charged with 60 lb. of mercury, during the treatment of 118 tons of pyrites, only 9 oz. 3 dwt. of gold were returned by the second dolly, whilst the

1st set of Tyrolese mills returned	..	..	34 oz. 16 dwt. of gold.
2nd " "	..	..	33 " 19 " "
3rd " "	..	..	27 " 19 " "
Total from the 3 sets	..	..	<u>96 oz. 14 dwt. of gold.</u>

That is, that of 105 oz. 17 dwt. which passed through the second dolly, it only saved 9 oz. 3 dwt. (8·5 per cent.), whilst the balance, viz. 96 oz. 14 dwt., was recovered by the three sets of Tyrolese mills, most of which, no doubt, would have been lost had the stuff been passed through even a third dolly.

Since by roasting pyrites lose $\frac{1}{3}$ rd in weight, the yield per ton of calcined ore must be computed at one-half more gold than that returned per ton of raw pyrites; that is, the yield of standard gold for 54 tons 4 cwt. 1 qr. 19½ lb. of raw pyrites having been at the rate of 2 oz. 4 dwt. 8 gr. per ton, the return per ton of calcined ore would be equal to 3 oz. 6 dwt. 12 gr.—an average which was not obtained by the calcining process, the return having been, as already stated, only 2 oz. 12 dwt. 14 gr., notwithstanding that the pyrites was clean, very well calcined, and was obtained from richer stone than that from which the 54 tons just mentioned, were obtained.

But, besides a larger percentage in gold, the new process saved 10½ oz. of fine silver, value 2*l.* 11*s.*, whilst by the calcining process hardly any silver is returned, at least of any commercial value. This in a great measure explains the higher standard of the gold obtained by the latter process.*

* The following are probably the reactions that take place during the calcining process in the roasting-furnace, and explain how a certain quantity of silver is lost, and the gold obtained from pyrites is of a higher standard if calcined previous to amalgamation in the Chilian mill, than that obtained from pyrites not roasted previous to amalgamation. During the first hours that the pyrites is exposed to an oxidising fire—roasted—the nascent sulphur, the presence of which is evidenced by the blue flames which constantly break through the charge, acts directly on the

It may be that in those localities where large quantities of pyrites are treated annually in a central establishment, to which the cost of transit from the mines is almost nominal ; where the cost of machinery, materials for building furnaces, stores, fuel, &c., is low ; where good and cheap labour can be obtained ; and where all the bye-products produced, such as sulphuric acid, white arsenic, sulphur, realgar, &c., can be readily sold at a profit ; that the calcining process and the subsequent amalgamation or chlorination or smelting of the roasted ore in conjunction with other ores would be the more remunerative process, and technically would perhaps be the more perfect ; but Rosales claims for his process the following advantages as regards the treatment of ordinary pyrites (iron- and arsenical-pyrites), viz. :—(1) That the pyrites at present allowed to run to waste can easily be treated in small parcels on the premises of any mining company situated in a mountainous district, or any other locality difficult of access ; as (2) no calcining-furnace is required, the erection of which demands costly materials, not easily to be had in such places ; (3) that the cost of manipulation is less than that of the system now in vogue, which requires that the ore should be previously calcined in a reverberatory furnace, although the bye-products are valueless ; (4) *a*, that a larger average yield of gold is obtained per ton of ore ; and *b*, that a certain quantity of silver is obtained, which is lost by the ordinary calcining process ; (5) that the loss of mercury is practically nil, whilst by the old process it varies from 1½ lb. to 2 lb. per ton of calcined ore.

The Chilian mill used by Rosales is an improvement upon the Peruvian *trapiche* and on the edge-runner mills of Real del Monte and Yendelaencina in Spain, in that the two rollers are provided on the outside face with a rim of white metal, and work on a revolving cast-iron pan fitted with a white-metal false bottom, made in sections, which can easily be replaced when worn out. The improvement in the Tyrolese mills consists in the addition of metallic surfaces coated with mercury to the faces of both runner and basin.

BISMUTH.—The occurrence of auriferous bismuth-ore is comparatively rare (see p. 102). Bismuth alloys readily with mercury, but does not seem to be so detrimental as antimony or arsenic. The sulphide causes the mercury to separate into “flour,” and gives rise to loss in the same way as other sulphides. The sulphide of bismuth found at Maldon,

finely-diffused particles of argentiferous gold, abstracting a certain portion of, if not all, the silver from them, forming sulphide of silver, which is subsequently, towards the end of the roasting process, altered into a soluble sulphate of silver, the temperature not being sufficient to change the sulphate of silver into an oxide, much less into metallic silver. On treating and amalgamating the thus roasted pyrites, grinding it wet in the pan of the Chilian mill, the soluble sulphate of silver is washed out and carried off with the refuse ; and thus occurs the loss of silver, whilst, on the other hand, its loss causes the gold obtained to be returned of a higher standard than it was when first charged into the furnace, in its original state.

though containing up to 20 per cent. of gold, does not give it up to mercury when triturated with the latter.

COBALT AND NICKEL.—Cosmo Newbery's process for antimony-ores, described on p. 580, embraces also the separation of cobalt and nickel as chlorides. When they are associated with sulphur, arsenic, or antimony, these latter are first removed by roasting; when the cobalt and nickel are free from these associations, the ores are first pulverised or calcined, then sufficient salt is added according to the assay, and the whole is roasted in the presence of aqueous vapour till the cobalt and nickel are converted into chlorides.

COPPER.—Copper is not injurious to the operations of recovering gold from pyrites, except in reducing the standard of the gold with which it is alloyed. On the other hand, by becoming amalgamated with mercury, it may actually assist in collecting the fine particles of metal, in the same way as amalgamated plates.

In dealing with auriferous ores containing copper, it should be borne in mind that in some cases the copper- and magnetic pyrites are almost free from gold, which is confined to the ordinary iron- and arsenical pyrites. In such circumstances, careful mechanical dressing of the ore might separate the auriferous non-cupreous from the cupreous non-auriferous portions, and each class could be treated alone for its particular metal. But in other instances, the association and relative proportions of the two kinds of pyrites do not admit of such a classification; and again, the cupreous pyrites are sometimes the most highly auriferous.

Under these latter conditions, metallurgical treatment must be resorted to. The most simple plan is to smelt the ore, and produce a copper regulus, which will contain most of the gold, sending the regulus to Swansea for final treatment. This plan was long followed by the owners of the Bethanga mines in Victoria. Their regulus contained 40 to 60 per cent. of copper, and 6 to 14 oz. of gold and 15 to 34 oz. of silver per ton. The results were not satisfactory; but much better returns were got by reducing the stuff to coarse copper, containing 98 per cent. of copper and 9 to 12 oz. of gold per ton.

Vautin's Smelting Process.—Vautin has introduced at Cobar, in New South Wales, an improved mode of treating sulphuret ores without the assistance of other ores. The *modus operandi* of the process is as follows:—A given quantity of pyrites or other mixtures of ores that will yield a regulus (say 23 to 10 per cent. of copper), is smelted in the ordinary way in the blast-furnace, and the regulus from the same is run into a molten mass, as hot and as rapidly as possible, through a short shoot into the converter, the blast for some having been turned on before the regulus is admitted. The result of the air being blown through the molten sulphides of copper and iron is that the sulphur is partly burned

into sulphuric acid gas and partly volatilised, and the iron is burned into the protoxide, and in that form combines with the silica, added for the purpose or contained in the ore, so as to form a slag, and in such condition it can be removed. It is claimed by Vautin that such reaction produces either a concentrated regulus of copper, say from 60 to 80 per cent., or even rough copper from 95 to 98 per cent. Both the sulphur and iron can be removed by one process, and with comparative ease, notwithstanding the fact that, under present methods of treating copper pyrites, these two elements in their removal form the greatest difficulty to deal with. When once the reactions are started in the converter, it is thought that raw sulphides might be added direct in the converter without previous smelting in the blast-furnace, as the heat produced during the oxidation of the iron and sulphur present in the first charge obtained from the blast-furnace is quite sufficient, not only to concentrate it as already described, but to smelt great portions of ore continuously.

Lixiviation with Hyposulphites.—Those ores which do not yield sufficient copper or lead for smelting, are poor in precious metals, and contain much antimony, arsenic, and sulphur, as is often the case in the Western States of America, lend themselves well to a lixiviation process by means of hyposulphite of soda or lime. The details of such a process were fully described by Prof. Egleston at the Roanoke meeting of the American Institute of Mining Engineers, and from the report of that meeting the following account is taken.

As compared with chlorination processes, it may be mentioned that whereas the salt consumed by them is low priced, the hyposulphite is high priced; but on the other hand, while the salt is used in quantity and absolutely lost, the hyposulphite really consumed is very little, most being recovered for re-use. Moreover, the consumption of water is small, the plant is simple and inexpensive, and ordinary labour suffices; but expert management is essential to prevent any irregularity in the roasting, leaching, or precipitating, whereby serious losses may easily be incurred.

A sample of the ores commonly dealt with by the process shows the following analysis, but in this instance the silver is unusually high:—

	per cent.		per cent.
Silicic acid	50·25	Sulphur	0·96
Iron	8·06	Carbonic acid	8·30
Zinc	7·62	Water	3·80
Lead	4·64	Loss and Oxygen	6·80
Arsenic	0·73	Alumina	trace
Antimony	1·35	Bismuth	
Silver	0·17	Copper	
Lime	4·92	Potassium	
Magnesia	2·40	Sodium	

The process consists of seven different operations :—(1) crushing the ore, (2) drying the ore, (3) roasting it with salt, (4) leaching out the base metals with water, (5) leaching with hyposulphite of soda, (6) precipitating the silver, (7) roasting the sulphide of silver and melting for bullion.

(1) *Crushing the ore.*—The ore is crushed so as to pass a 15 to 20 mesh screen ; 30 mesh screens were first used, but did not discharge as well, and the roasting and leaching are better and quicker done on coarse ore. A general rule is that the ore must be crushed just as coarse as is consistent with perfect chlorination with salt in the furnace.

(2) *Drying.*—The driers are revolving wrought-iron cylinders, 20 ft. long by 4 ft. in diameter at one end and 3 ft. at the other, known as "Pacific" driers. The ironwork for the driers weighs about 10 tons ; they are not lined. The flame from the fireplace runs directly through them, the ore being automatically fed at one end and dumped out into cars at the other end without manipulation. The driers need not be heated by a fireplace of their own, as the flames from the Brückner's cylinders might be made to pass through them and then into the dust chambers, thus utilising a large amount of waste heat. Drying-floors, made by passing the waste heat through flues covered with cast-iron plates, are used in some works. This saves the fuel used in the driers ; but is more than compensated for by the labour required. The dry ore is carried to a bin, from which it passes over a 15-mesh screen, and falls through a shoot in which are magnets to catch any pieces of iron which may have accidentally got into the ore. It then passes through two sets of Krom rolls, 16 in. by 24 in., from which all that passes through the screens is elevated to a storage bin.

(3) *Roasting with salt.*—From the bin the ore descends through a shoot into cars standing on a scale, where it is weighed. The contents of the cars are dumped into a hopper above the Brückner's cylinders, each charge being carefully weighed. The moment the hopper is discharged into the cylinder beneath another charge is put in. The salt is measured in soap boxes which contain about 80 lb. each, and is mixed with the ore either in the driers or in the hoppers. Formerly 5 per cent. of salt was mixed with the ore ; now only 3 per cent. is used. The best place to add the salt is in the driers ; the incorporation resulting from the movement there and in the rolls considerably reduces the consumption of salt. An ingenious mixing machine, into which the ore and salt are charged, has produced great economy in the use of salt and better subsequent working. At first 2 per cent. of iron pyrites was mixed with the ore in order to ensure proper roasting ; now none is used, but where there is a large amount of base metals this addition will be necessary.

Roasting may be done in any kind of furnace. If a reverberatory furnace the hearth should be arranged in three steps, so that the ore in passing from one to the other falls a distance of 4 to 5 ft. A Stetefeldt furnace can be easily transported, but is more expensive to build than a reverberatory furnace, which can always be easily adapted to any kind of fuel. In Brückner's cylinders, which are 7 ft. diameter by 19 ft. long, and hold a charge of about 5 tons, the fireplace is on a prolongation of the axis of the furnace. The cylinder is driven by friction rollers, of which there are three sets, replacing the gear-wheel round the body of the cylinder of the older form. The work is continuous, the furnace never being allowed to become cool. As soon as a charge has been treated, a fresh charge is immediately put in. To introduce the charge, the man-hole is brought under the hopper and its valve is drawn. It is then replaced and the cylinder is set revolving 2 or 3 turns a minute. If the amount of sulphur contained in the ores is exceedingly small, salt in very small quantity, if it has not already been added in the driers, may be introduced at once. When ores containing a large amount of sulphur are used, a careful roasting at a low temperature must precede the chlorination with salt, steam at a low pressure being introduced for the purpose not only of getting rid of the last trace of sulphur, arsenic, and antimony, but also to decompose the chlorides of the volatile base metals, the nascent chlorine thus given off acting very energetically on the silver. When the ore contains base metals which it is desirable to save, this roasting must be done with great care, and the value of the base metals separated must compensate for the extra expense in fuel owing to the use of steam in the roasting. If lead is present, the roasting must be done at a low temperature, for as the compounds of lead are easily fusible, there might be danger of agglomeration; or, if the temperature is high and silica is present also, a silicate of lead might be formed which would prevent the solution of the silver. Special care must be taken in such a case to transform all the lead into chloride, as this is soluble in hot water, while the sulphate is not.

The roasting lasts 8-11 hours, depending on how the charge works. When it is finished, the man-hole is opened without stopping the cylinder, which in its rotation discharges the ore, so that it falls into pits cut out of the rock in the foundation, just underneath the cylinders, where it is allowed to remain about 9 hours, until it is ready to go to the cooling floor. A considerable amount of chlorination takes place in the pit after the charge leaves the furnace, so that the time of waiting is not lost. It is red hot when it falls into the pits, but cools sufficiently to be drawn off into cars after that time. Occasionally the ore, when for any reason there is a stoppage, remains for 2 or 3 days in this bin, and is still hot when drawn. Generally, however, it is drawn out on the

cooling floor as soon as possible, where it is at once moistened with water to keep down the dust. Sometimes the ore is put into the tub so hot that the water boils, but this is not usual. The ore is generally cold enough not to make any appreciable difference in the temperature of the water. When the ore is one which is not habitually treated, a sample is drawn through the fireplace in order to test the chlorination. When, however, the ore is that generally treated, the workmen recognise that it is finished by its rolling about in the furnace with a sluggish motion somewhat like that of damp sugar.

When about half the charge of the furnace has run out, a long iron spoon is run underneath and filled, and its contents are assayed. One sample is used for a chlorination test, which is made at once for each charge, each assay sample being marked with the number of the charge and of the furnace or the cylinder; the other is thrown into a large iron car to be afterwards used in making the general assay to ascertain the amount of silver contained in the ore. When the charge of ore from the furnace is more than the tub will hold, the residue is put into a heap, and when enough of it has accumulated to fill a tub, it is leached and called a "mixed charge." The assay for chlorination is made on the sample taken from the charge: 15 oz. are weighed out and put into a funnel with a proper filter; hyposulphite from a tank above is let on to it and allowed to run 15–20 minutes, the filtrate being collected and sent to the leaching tubs. When no silver is dissolved, the assay is washed and dried, and a fusion assay is made of the tails.

A considerable amount of chloride of silver and of dust containing silver is volatilised or mechanically carried off from both the Brückner's cylinders and the driers, necessitating dust-chambers connecting with both. The flues have a down-take, which communicates with a large chamber by an arched opening, which is quite small. The dust accumulates in considerable quantities at the bottom of the down-take. This material and that which comes from the dust-chambers is passed through the Brückner cylinders and mixed with the charges. Fresh salt is added to it, and it is put through the tubs. Sometimes, when large quantities are on hand, it is treated separately.

(4) Leaching the base metals with water.—The leaching-tubs are 6 ft. diameter and 3 ft. deep, the depth being regulated according to the facility with which the water will filter through the ore, and also to the ease with which a man can throw out the tails. If the diameter of the tubs is increased to 12 ft. a better result is obtained. The tubs are set on the floor together in pairs, almost touching each other, with a railway close at one side and an easy passage-way between each pair of tubs. The wagon which receives the tails comes just above the top of the tubs, so that a man standing in them can easily throw the tails out into the

wagon. Permanently fixed above the tubs are long narrow hoppers, divided into two compartments, and having at their bottom a slit-valve so that the ore may be discharged from each compartment in little piles over all parts of the bottom of the tub. Each tub may have its own iron hopper, or one hopper may answer for several tubs, by using flexible pipes. Stop-cocks or plugs are apt to get out of order and to leak. A rubber pipe can always be securely fastened to the bottom or side of the tub, and when not in use can be hung up out of the way. The ore is passed from the cooling-floor to the vats as rapidly as it can be handled, each charge of ore being drawn from the furnace-bins to the cooling-floor as soon as the latter is empty. The cars from the cooling-floor run over the top of the hoppers and dump their contents into them, which from there fall into the tubs when the slit-valve is opened, and the ore is afterwards evenly distributed over the bottom of the tub with a hoc. The charge could as easily be put into the tubs from a car running on a railroad at a sufficient height above the tub not to inconvenience the workmen in discharging it, and arranged with a valve like the hopper, or it could be dumped into the vat from the side.

The bottom of the tubs is covered with 4 or 5 wooden slats, 3 in. by 1 in., which do not touch the sides. Over these, at right angles, are arranged other slats 1-1½ in. apart. Gunny-sacking, which is wet from the previous charge, or is wetted purposely when a new filter is to be put in, is placed over these. It is brought close up against the sides of the vat, so that no ore will pass, by means of a hoop, which fastens it securely. The ore falls from the hopper upon this canvas and rises to within 2-3 in. of the top of the tub. When the ore is once in the tub, special care is taken that it shall not be disturbed in any way. Once ready for the water, it is allowed to remain without being disturbed until the tails are ready to be discharged. Any interference with the arrangement in the tub increases the difficulty of leaching. The simple pushing of a stick two or three times down through the ore may delay the leaching several hours.

In the bottom of the tub are 2 rubber pipes, about 1½ in. diameter, one for introducing the water, and the other for discharging it. When hot water is used for leaching out the base metals, the ore is always wetted from the bottom, the water being introduced through the pipe and coming up through the ore, as this method of moistening causes the ore to cake less and the solutions to percolate much more easily. As soon as the water completely covers the ore, the supply is cut off. After remaining there for a very short time, the discharge-pipe, which has up to this time been hung up higher than the top of the tub, is lowered, and the water is allowed to run out; the leaching water is added at the same time. This water is permitted to flow away entirely, or is collected,

according to the quantity of water available. The water used for the base-metal leaching may be hot or cold. If the ore contains a large amount of lead, which has been converted into chloride, it will be best to leach with cold water until the larger part of the lead and the surplus salt have been dissolved out. It is then leached with hot water, cooling the ore, however, with cold water before the hyposulphite is added, in order to prevent too great an extraction of the base metals with the silver. When the ore filters slowly, it will be found best to heat the water, which will not only make it filter more easily, but will dissolve the base metals more rapidly. When the charge coming warm from the cooling-floor is introduced into the tub, the water grows warm; then, unless the ores are very pure, or impure ores have been leached with hot water, the charge must be cooled with cold water before introducing the hyposulphite, or the bullion would be much more impure.

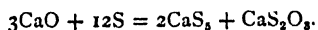
As the water introduced from the bottom subsides, a very thin crust is formed upon the top of the charge, which is carefully removed and put by itself until sufficient accumulates to be treated. This material contains all the silver which was dissolved by the excess of salt or other chlorides in the ore, and which would have been lost if the hot water had been introduced at the top; the amount is all the larger if the solutions are hot, or if the excess of salt is large, as a hot brine dissolves more silver according as it is hot and saturated, while a cold one dissolves hardly any. The dilution of the liquor with water precipitates part of the silver near the top, and distributes the rest of it through the ore, so that but little is lost in bottom leaching. After the top crust has been removed, clear cold water is allowed to run in at the top, the quantity being regulated so that the inflowing water will just be equal to the quantity discharged, in order to leach out the base chlorides which are soluble. Any salt in excess, or any which has not been decomposed in the roasting, will be dissolved at the same time. When the ore contains but little base metal, there is no advantage in using hot water. But when the ore contains a great excess of lead or antimony, introducing the water from below is not sufficient, as the chlorides of these metals also are precipitated from the solution by dilution with water. In such a case the water is introduced from the top, and the liquid is allowed to flow from several tanks into compartments filled with shavings, so as to get a large amount of surface. Clear water is allowed to flow into these compartments, till the liquor is dilute. The moment the fresh water touches the stream containing the chlorides, which was formerly clear, it becomes cloudy, and then precipitates the chlorides. Flowing over so large a surface, and being obliged to pass under one compartment and over the other, the chlorides of silver, lead, and antimony deposit on the shavings, and can be dissolved from them with

hyposulphite, and the solution put with the other silver solutions. The water running off contains the zinc, copper, and iron. In Mexico, the washing is repeated ; the ore is discharged from the first tub into a second ; the first washing lasts 4 hours and the second a little less ; even when no metal salts are found in the first wash-water some are found in that of the second, and only when nothing is precipitated from the second washing is the leaching with hyposulphite begun.

When the water flowing from the tubs no longer gives a precipitate on testing with calcium sulphide, the base metal chlorides are removed. When there is plenty of water, it is allowed to run to waste, if the ore contains no nickel, or cobalt, or other metals worth collecting. When water is scarce, or the base metals are valuable, it is all collected in the vats. The base metal chlorides are precipitated with calcium sulphide, and the water is used over again. It takes a quantity of water equivalent to 3 tubfuls to leach out the base metal chlorides.

(5) Leaching with Hyposulphite of Soda.—The washed ore is leached with a cold solution of hyposulphite of soda, its strength depending on the richness of the ore and the quantity of base metals present. If the ore is very rich, and but little base metal is present, it may be used very strong, and even warm ; but when base metals are present the solution is usually made weak and used cold. The quantity used may be from 1 lb. to 8 gal. of water, to $\frac{1}{2}$ or $\frac{3}{4}$ lb. to the gallon. The hyposulphite is usually purchased in small kegs of 50–60 lb. each ; costing 5 to 6 cents a lb. it is generally cheaper to purchase it than to make it. The liquors increase in quantity as the solutions are constantly being regenerated, so that but small additions have to be made, and this only to keep up the strength of the solution, as the liquors are constantly being diluted. The hyposulphite is likely to become impure when the ores are not properly leached with water to dissolve out the sulphate of soda and chloride of sodium ; it is generally the practice then to regenerate it by spreading wood ashes over the ore and leaching through that, the alkalis in the ashes taking up the sulphates and the excess of chlorides. By the constant use of the polysulphide of calcium, it gradually becomes converted into hyposulphite of lime. It is sometimes desirable, especially when the ore contains gold, to use hyposulphite of lime, especially as the polysulphide of calcium which is used for the precipitation of the silver is the first step in the process of manufacturing it. This is done by boiling $1\frac{1}{2}$ parts of the purest freshly slaked lime that can be had with 1 part of crushed brimstone. Flowers of sulphur does not answer so well as the brimstone, which can be more readily obtained and is more easily manipulated. If the lime is not very pure, it may be desirable to use 2 parts of lime to 1 of sulphur. When sulphur is scarce and high-priced, it is sometimes collected from

roasting the sulphide of silver. The water is first boiled with steam, the lime is added and well stirred, and the sulphur is then introduced. Sufficient water must be added to keep the mass liquid, but not enough to allow the solution of polysulphide to become too much diluted. The boiling is kept up 3-4 hours. During this time polysulphide of lime, mixed with hyposulphite of lime, is formed, as shown by the formula :—

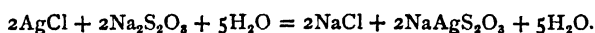


This operation requires some care. If lime is added in excess, insoluble compounds are apt to form, e.g. bisulphide of lime, which forms reddish-yellow crystals. An excess of sulphur does no harm, and may be useful in the next operation. The liquid is allowed to settle and the clear liquid is decanted. It is desirable that this solution should be at 8°-10° Baumé, and only sufficient water is added to bring it up to that strength. The liquid is now treated with sulphurous acid, made by boiling charcoal broken to about the size of a grain of corn, with sulphuric acid of about 1·80° B., enough being used to make a pasty mass. This is done in an iron retort, adding the acid to the charcoal as it is necessary. There is no need of purifying the gas. When the yellow colour of the liquid disappears, the sulphide has been converted into hyposulphite. This is tested with a dilute solution of chloride of silver. So long as there is any precipitate or cloudiness, the passage of the sulphurous acid must be continued; when there is none, the solution is ready. The residue in the tub is next treated with the same amount of water as before, and the operation is commenced over again. If the resulting liquor is not more than 3°-4° B., it is too weak, and is kept to treat the next charge of lime. A simpler process is sometimes used. The residues left in the tub after the clear sulphide liquid is decanted are simply shovelled out and left for 3-4 days exposed to the sun. They are then leached with water, which extracts the hyposulphite of lime, and thrown away. This method is very rough, as the oxidation is necessarily imperfect, but for some situations it is much more economical than using sulphurous acid.

The sulphide of calcium made in the first stage of the process is the material used for precipitation. It should not be less than 6° B. In some works the polysulphide of sodium, made by boiling soda with sulphur and treating it with sulphurous acid to make hyposulphite of soda, is used. This reagent does not precipitate the silver so rapidly, nor so well, and is not so easily washed. For this reason the hyposulphite of lime is preferred. Sometimes sulphuretted hydrogen, made by melting paraffin and flowers of sulphur together, is used for the precipitation. This is much more disagreeable. Where hyposulphite of soda is used, the constant addition of lime transforms the solution

gradually into hyposulphite of lime. The strength of this solution is constantly checked by hydrometer.

When the ore is very impure, it is generally best to keep the solution at about $\frac{1}{2}^{\circ}$ B. Before turning the solution into the precipitating vats, care must be taken to see that the water used for leaching out the base metals has been displaced. This is easily done by tasting, as the hyposulphite of soda and silver is intensely sweet; or better by testing the liquor flowing from the tanks with sulphide of calcium. As soon as the least turbidity is shown, it is time to catch the liquor, as the hyposulphite is acting. This hyposulphite liquor is allowed to run through as long as it has a sweetish taste. The reaction which takes place is:



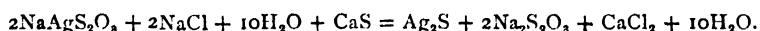
This hyposulphite of soda and silver is exceedingly soluble. The quantity of hyposulphite, and also the time required for leaching, will depend on the richness of the ore, more time being required for a rich than for a poor one. Ores are rarely treated that require a longer time than 20–30 hours. An ore containing \$300–\$400 (60%–100%) a ton, will be perfectly leached in 12–15 hours.

As soon as the hyposulphite ceases to taste sweet, the solution is tested with sulphide of calcium to ascertain whether the ore is exhausted of silver. The tester carries a small bottle of the sulphide solution, in which he has a stick. He takes a tumblerful of the liquid running out and lets a drop or two of the sulphide fall into the liquor in the glass. If there is a precipitation of sulphide of silver, the lixiviation is continued. If there is no precipitate, and the liquor becomes only slightly discoloured, he puts in some of the liquor containing the silver solution to ascertain whether there is an excess of sulphide of calcium. In this case the hyposulphite solution is discontinued and the excess of hyposulphite must be washed out with cold water. The exhaustion of the hyposulphite is distinguished by the taste. This last liquor, as it contains nothing, goes to the fresh-water tanks.

It takes 18–48 hours to charge, leach, and discharge the ore, the time depending on the way the roasting has been done. Exceptionally a charge will take a longer time, sometimes as much as 5 or 6 days. In Mexico, the leach liquor is divided into two parts, as that coming off first contains much less of the sub-sulphates and oxychlorides of the base metals than the last, and consequently produces a purer bullion. When all the product must be cupelled with lead, this is not necessary. As soon as the water ceases to flow, a probe sample is taken from three points in each tub, and they are sent to the assay office to see whether

there is any silver left. If the ore assays 40–50 oz., the tails should assay only about 4 oz. If the assays show that the silver is down to 4–6 oz., the tub is discharged; if not, the ore has to be re-roasted and treated with the hyposulphite again.

(6) Precipitation of the Silver.—The hyposulphite liquor containing the silver is run directly from the leaching vats to the precipitating tanks, which are 8 ft. diameter, and 12 ft. deep. All the solutions, except the silver liquors, are carried about the mill through wooden launders, carefully jointed with tar, and also painted on the inside with it. This is an excellent method. Iron pipes decompose the solutions and sometimes precipitate the silver. Iron pipes lined with tar are better, especially in those mills whose work is not continuous. These might be ordinary steam-pipes, 5 in. diameter, screwed together. The silver liquors from all the leaching tubs are run into one precipitation vat until it is filled to within 15–20 in. of the top; the flow is then turned into the next vat. When this is done, the silver is thrown down at once, sulphide of calcium being added until no precipitation takes place. The solution is agitated for some minutes to make a thorough mixture, and is then allowed to settle, and the clear liquor which is reconverted into hyposulphite of lime and soda by the sulphide of calcium is drawn off into a receiving tank on a lower level, and from there forced into a tank on the top floor, from which all the solution used runs. The reaction which takes place is



This solution varies from 25°–30° B. to much weaker.

Where various ores are treated, it requires a little time to ascertain just the quantity of sulphide to add, but when the same ores are constantly being used, it is simply measured in from a pail. There is no danger of loss, as the liquors are carefully tested. Care must be taken not to have the sulphide in excess. Only that portion of the sulphide which precipitates the silver is converted into hyposulphite of lime; any excess of sulphide would cause a precipitation of the silver already dissolved in the solution-tanks as sulphide, which would not be dissolved by the hyposulphite and would consequently be lost in the tails. If there is an excess of sulphide, some of the silver liquor must be added to neutralise it. As soon as the clear liquors are run off, other liquors are run in and precipitated, clarified, and so on, for about 2 weeks; at the end of that time the tanks must be discharged. There are precipitating tubs for the sulphide of silver and for the precipitation of the base metals with the same sulphide when water is scarce. Much trouble has been experienced in the various works in pumping these solutions. Iron pumps do not work well, and steam pressure has a great many incon-

veniences. The simplest way is to use an ordinary lift pump made with a wooden cylinder lined with hard rubber with a hard rubber plunger.

Every 15 days the accumulated sulphide of silver is collected. The arrangement is made to have the whole mill cleaned up at one time. To do this the sulphide in the bottom of the tanks is stirred vigorously until the material is in the state of a thin mud. It is then drawn off from the bottom of the tank and run over cloth filters. These filters consist of a series of frames about $2\frac{1}{2}$ ft. square, over which sacking is securely fastened, built on a table, which is surrounded by a rim 3 in. high. Underneath the table is an inclined trough which carries all the liquors to the collecting vat. The turbid liquor from the precipitating vat is run on to this table, and flows from one sacking to the other. The solution drains through, and the pasty sulphide remains on the sacking. This operation is constantly repeated until the sulphide of silver has accumulated in considerable quantity on the sacking. It is then washed with clear water to remove the excess of sulphide of calcium until it is perfectly sweet. It is then left to dry. In drying, it cracks in every direction. It is shovelled into a car and carried to the reverberatory furnace to dry and roast. As there is a considerable amount of water in the pasty mass, it is in some works collected in canvas bags, and the water is squeezed out with a press before it goes to the furnace.

(7) Roasting the Sulphide of Silver and Melting for Bullion.—The sulphide of silver is now ready to be dried in a reverberatory furnace, as it still contains some water. The heat must be quite low at first, only just enough being used to burn off the sulphur. It can only be increased very slowly at first for fear of melting the sulphides. It is roasted until most of the sulphur is driven off, and the heat is then raised to as high a temperature as possible without melting. When sulphur is scarce, the sulphide is heated in a retort, and the sulphur is condensed to be used over again. Aaron proposes to take the freshly precipitated sulphide and boil it, adding freshly slaked lime in small quantities at a time to the liquor, which is kept constantly agitated. When this is carefully done, the polysulphide of calcium will be formed much more quickly than by the direct action of the sulphur, and most of the sulphur in the sulphide of silver will be regained. The liquor is decanted, and the residue is dried and roasted. After roasting, it is a gray mass composed almost entirely of metallic silver. It is cupelled with the addition of lead in an English cupel. The litharges are rich in silver, and are all reduced to form the lead for the next cupellation. From the cupels it is cast into bricks, and is between 800 and 900 fine. No attempt is made to make it finer, as it is found cheaper to sell it than it is to refine it. If the ore contains a large quantity of base metals, particularly lead, these would be only partially affected by amalgamation, whereas most of the

lead contained in the ore will be found in the bullion made by leaching. The reason for this is that while all that portion of the lead which is in the form of chloride may be leached out, and, by the judicious arrangement described, the other metals can be separated, none of the lead present as sulphate will be attacked by the water, while most of it will be dissolved by the hyposulphite and will be precipitated by the sulphide, thus using a larger amount of the reagent, and being a direct source of loss. This, however, except in very poor ores, is not a grave inconvenience. In other respects the bullion is likely to be purer. The loss is practically very small. The outlay for chemicals is also very small, as they are constantly regenerated, and the same water can be used over and over again. The amount of necessary loss in the tails will depend, other things being equal, on the price of labour. It is very easy to ascertain this by making a daily clean up, or making a time assay, to ascertain exactly how much is being extracted in a given time. The assay shows that 85 per cent. of the silver in the chloridised ore is saved. From 50 to 60 tons are easily treated in the 24 hours, at the Bertrand mill, where the labour employed per 24 hours is as follows:—

			\$	c.	\$	c.	£	s.	d.
Roasters—furnace men	..	..	2	@ 4 00	8	00	1	12	0
„ —feeders	..	..	2	@ 3 00	6	00	1	4	0
Driers and rollers	..	..	10	@ 3 00	30	00	6	0	0
Cooling-floor	..	..	6	@ 3 00	18	00	3	12	0
Leaching-floor, Chinamen	..	..	2	@ 2 50	5	00	1	0	0
„ „	..	..	3	@ 1 50	4	50	0	18	0
Rock breakers	..	..	2	@ 3 00	6	00	1	4	0
Flue cleaners	..	..	4	@ 1 50	6	00	1	4	0
Blacksmith	..	..	1	@ 6 00	6	00	1	4	0
„ helper	..	..	1	@ 3 00	3	00	0	12	0
Carpenters	..	..	2	@ 5 00	10	00	2	0	0
Engineer	..	..	1	@ 5 00	5	00	1	0	0
„	..	..	1	@ 4 00	4	00	0	16	0
Fireman	..	..	1	@ 2 50	2	50	0	10	0
Assayer ..	..	..	1	@ 5 00	5	00	1	0	0
Lamp cleaner	..	..	1	@ 1 50	1	50	0	6	0
Foreman	..	..	1	@ 6 00	6	00	1	4	0
„	..	..	1	@ 5 00	5	00	1	0	0
Office helper ..	..	..	1	@ 1 50	1	50	0	6	0
General helper	..	..	1	@ 3 00	3	00	0	12	0
Watchman	..	..	1	@ 3 00	3	00	0	12	0
Woodmen, Chinamen	..	..	3	@ 1 50	4	50	0	18	0
„	..	..	1	@ 3 00	3	00	0	12	0
					\$146	50	£29	6	0

All the charging, discharging, and leaching are done by 4 Chinamen and 3 Irishmen. The total cost of milling the ore is \$6.50 (26s.) per ton. The cost of mining and delivering the ore at the mill is about

If this precipitates the gold, it is certain that there is an excess of precipitating reagent retained in the vat, and that no more needs to be added. It has sometimes happened that after the leach water showed no trace of gold, and the discharge was stopped for a few minutes and again started, there was a precipitate for a few minutes, which would soon stop, and that this might be repeated several times. This is owing to the unequal percolation of the water, and unless guarded against might be a source of considerable loss.

The precipitating material employed is protosulphate of iron, which is prepared at the works, and is generally very acid. It is made by pouring 1 gal. sulphuric acid into a tub 4 ft. diameter, 2 ft. 6 in. high, filled with scrap iron covered with water, so that there is always an excess of iron: 1 gal. acid makes a tubful of solution of the requisite strength. This solution is siphoned off into a vat 2 ft. 6 in. diameter at top and 3 ft. at bottom and 3 ft. high, and is drawn from thence to be used in the precipitation tanks by a rubber hose, which is hung up when not in use. When green vitriol (iron sulphate) can be had cheap, a saturated solution of it is made, which is filtered before being used, but generally iron and sulphuric acid are cheaper.

The gold in the precipitating tanks is generally allowed to settle for 12–16 hours, or at least over night. Sometimes it is necessary to allow it to remain 24–26 hours, for the gold is often so exceedingly fine that a slight purple colour will easily be distinguished for a long time. So long as it can be seen, there is gold in suspension. The time usually depends on the heat of the weather, taking much longer in winter than in summer. When the clear supernatant liquor shows no gold on testing, the upper part of the liquor is removed with a siphon, or by tapping the tank at different levels, as little liquid as possible being left on the precipitate. As both these methods require watching, the plan generally adopted is to use a flat piece of wood *a*, attached to a rubber hose *b*, Fig. 250. This is done in such a way as to keep the mouth of the pipe always under the level of the water, until there would be danger of the gold on the bottom being disturbed, when the width of the float prevents the liquor from being drawn off any further. When not in use, the pipe and float are hung up, as at *c*. This method requires no watching. All the liquor flowing off after the precipitation of the gold, is collected in large tanks and allowed to settle until place is required for other liquor, when it is drawn off. Very little gold is ever caught in this tank, so that it is only cleaned at long intervals, generally only when the yearly inventory is made.

It is never certain that when liquor appears clear to the eye no fine gold is in suspension, so it is always safer to allow this liquor to flow through a filter of sawdust or sand, as at *k*, Fig. 253, before running to

waste. If sawdust is used, it is burned after a time and the gold is recovered from the ashes ; if sand is used, when it is taken up it is chlorinated like the ore. Both methods are simple and effective, but do not equal precipitation of gold by charcoal.

The precipitated gold, unless the ores are very rich, is allowed to collect for several operations. It is then dipped off from the bottom of the tank with a clean porcelain or enamelled dish, and the bottom and sides of the vat are thoroughly brushed with a small brush and washed with the least possible quantity of water, the final residuum being washed out with the hose from the bottom stop-cock of the vat, settled, and filtered. The precipitation tanks are kept covered and locked, except when it is necessary to have them opened to remove the gold or to clean them. When they are not coated with the waterproof paint they absorb a certain amount of the solution, so that when they become old or leaky they are burned to recover the gold from the ashes. They, however, last a very long time.

The precipitate obtained is treated with a mixture of sulphuric acid and salt to free it from iron salts. It is first red, and changes afterwards to a brown gold colour. The purified precipitate is placed in a cloth and strained to dry it, and is then squeezed by twisting to make it as dry as possible. When it is moderately pure, it will fall after squeezing with a touch of the finger, but when filled with impurities it will not. These cloth filters are always kept under water while not in use. When worn out, they are dried, a little nitre is sprinkled over them, they are burned, and the gold is recovered from the ashes. The gold is dried on a pan, but removed before it is completely dry, in order to avoid loss of fine dust which is very mobile. Sometimes the precaution of using the cloth filter is not considered necessary. The gold is collected on a filter made of punched sheet iron covered with two or three thicknesses of ordinary filter paper, and washed with water until it is sweet. It is then dried over a low fire and melted in a cast-iron or graphite retort with salt, borax, and nitre. The gold is generally 800 to 900 fine, sometimes 960, the impurities consisting for the most part of iron and lead ; \$175-400 (35-80%), and sometimes more, are taken from the tank at a time ; 95 per cent. of the gold is extracted.

This method of precipitation is difficult, because the fine gold remains so long in suspension, a purple colour being seen in it for several days, and because all the wash water must be kept, on account of the small quantity of gold it is liable to contain. The solutions of glue and tannin which have been successfully used in some of the works in Europe, to collect the precipitates of silver, do not seem to have been satisfactorily used upon gold. In order to prevent this loss Davis proposes to precipitate the gold from solutions by means of charcoal. According to him it

requires 240 gr. charcoal to precipitate $19\frac{1}{4}$ gr. gold, or about 100 lb. charcoal to precipitate \$2450. To effect the precipitation, the gold solution, after settling, is made to flow through charcoal that has been crushed to pass a screen of 25-50 meshes. This is placed in barrels having a quartz filter on the bottom; 100 lb. of the crushed charcoal fill the barrel to a height of 28 in. It is desirable that there should be two barrels, the gold solution being introduced into the first at the rate of 20 gal. per hour; the solution passes through it, flowing into the second at the same rate. This is done in order to be certain to catch any gold not precipitated in the first. The liquor from here goes to a collecting tank, perfectly freed from gold, in order to precipitate any other metal that may be contained in the solution. Copper can be collected as cement copper, and the remaining liquid allowed to flow to waste. After 960 gal. of the gold solution had passed over the charcoal, no traces of gold could be detected in the solution leaving the barrel.

At Deloro, the lime contained in the ore gives a voluminous precipitate with the iron sulphate. To avoid this the lime was for a time precipitated with sulphuric acid, but this was found too tedious and had to be abandoned. With charcoal, precipitation of the gold was complete. Lime, after the liquor passed through the charcoal, undergoes such a chemical change that it is no longer precipitated, either by iron sulphate or sulphuric acid, although it is by oxalate of ammonia. When a certain volume of the solution has passed through the charcoal, it is removed in wooden trays, dried in the air, and burned in a muffle furnace. The ash is treated with muriatic acid and washed with water to remove any soluble salts; then dried, and melted in a crucible with fluxes, and usually gives very fine gold. To ascertain the exact precipitating value of the charcoal filter at different heights in the barrel, 5400 gal. of one solution were allowed to pass through a bed of charcoal 20 in. thick. The filter was divided into 4 equal parts of 5 in. each, and these were separately burned. The upper strata gave 2 oz. 95 gr., the second 2 oz. 99 gr., the third 1 oz. 222 gr., and the last 1 oz. 62 gr. The precipitation and collection of the gold is a weak and tedious part of the process. It has been proposed to precipitate the gold with H_2S , made by melting paraffin and sulphur together and forcing the liquor through a filter press. Many advantages are connected with the use of the filter press in separating precipitates.

The ore remaining in the tubs contains the greater part of the silver converted into chloride, which is insoluble in water. It is left in the tubs generally to drain for 24 hours; then removed to other vats or left in the same, and leached with hyposulphite solution, by which 60 per cent. of the silver is extracted. It is better, generally, to chlorinate the ore first, and leach for silver afterwards. All the work of leaching, impregnation,

solution, precipitation, and fusion is done in the daytime; only the roasting is continued during the night.

At the works of the Plymouth Company, they treat 100 tons of concentrates per month of 30 days. The leaching occupies only 24 days in the month. The expenses are :

<i>Roasting.</i>				\$	c.	\$	c.
3 men at 2.50 dols. a day for 30 days	..	..	..	225	00		
1½ cords wood at 4.25 dols.	..	..	..	223	13		
54 lb. salt at ¼ c.	..	..	..	12	15		
						460	28
<i>Chlorine.</i>							
60 lb. manganese per day at 47 dols. per ton	..	..	..	33	84		
68 lb. salt per day at 15 dols.	..	..	..	12	24		
120 lb. acid per day at 60 dols.	..	..	..	86	40		
						132	48
<i>Leaching.</i>							
40 lb. sulphuric acid for 24 days	..	..	..	57	60		
40 lb. sulphate of iron for 24 days	..	..	..				
1 leacher at 5.50 dols. for 30 days	..	..	..	165	00		
Salary of foreman	..	..	..	125	00		
						347	60
						940	36

Or per ton of concentrates, 9.403 dols.

At the Providence Mine are 2 roasting furnaces of the capacity of 9 tons in 24 hours. Each furnace requires 1 cord of wood in that time. The total cost is :

	\$	c.	£	s.	d.
1 foreman	3	00	0	12	0
1 white labourer	9	25	1	17	0
5 Chinamen at 1.50 dols.	7	50	1	10	0
2 cords wood at 5 dols.	10	00	2	0	0
29 lb. binoxide manganese at 2½ cents	0	80	0	3	4
260 lb. salt at 1 cent	2	60	0	10	6
216 lb. sulphuric acid at 2 cents	4	32	0	17	6
Lime, sulphur, and calcium hyposulphite	0	30	0	1	3
Illuminating	0	20	0	0	10
Extras	1	00	0	4	0
Total	38	97	7	16	4

This makes the cost of treatment per ton of sulphurets \$3.55 (14s. 9d.) when the works are run at full capacity. But the ore contains about 7 per cent. of sulphurets, or 4½ tons in the 62 tons milled daily. This quantity of sulphurets does not keep the two furnaces running at full capacity, but both are in continual operation. Most of the expenses remain the same whether running at full capacity or not; the actual

cost, therefore, figured on a working basis of $4\frac{1}{2}$ tons daily capacity, approximates as follows :

	\$	c.	£	s.	d.
Labour	12	75	2	13	2
2 cords wood at 5.00 dols.	10	00	2	1	7
14 lb. binoxide manganese at $2\frac{3}{4}$ cents	0	38	0	1	7
126 lb. salt at 1 cent	1	26	0	5	3
104 lb. sulphuric acid at 2 cents	2	08	0	8	8
Lime, sulphur, and calcium hyposulphite	0	15	0	0	8
Illuminating	0	20	0	0	10
Extras	0	50	0	2	1
Total per day	27	32	5	13	10
Adding to this the cost for milling per day	57	50	11	19	7
The total outlay per day equals	84	82	17	13	5

Or \$1.37 (5s. 8d.) per ton for extracting the gold and silver from the ore.

This estimate makes no allowance for the expenses of general supervision, interest on first cost, and gradual deterioration. The conditions of treatment in these works are, however, very special, and can hardly be considered as a basis for the general cost of treatment elsewhere.

It will thus be seen that the cost of roasting is by far the greatest expense in the treatment of the concentrates, it being more than twice that of the chlorination. When a steady supply of ore can be had, the cost of treatment will always be less than when the works are often idle for want of ore. By adopting the Stetefeldt furnace or Brückner cylinders, or some other mechanical furnace for the finishing roast, it is probable that a considerable expense might be saved. The margin of profit was formerly very large, for at the Lincoln Mine the usual charge for custom ores was \$25 (5*l.*) per ton. The difficulty seems to be the frequent enforced idleness of the works from want of ore.

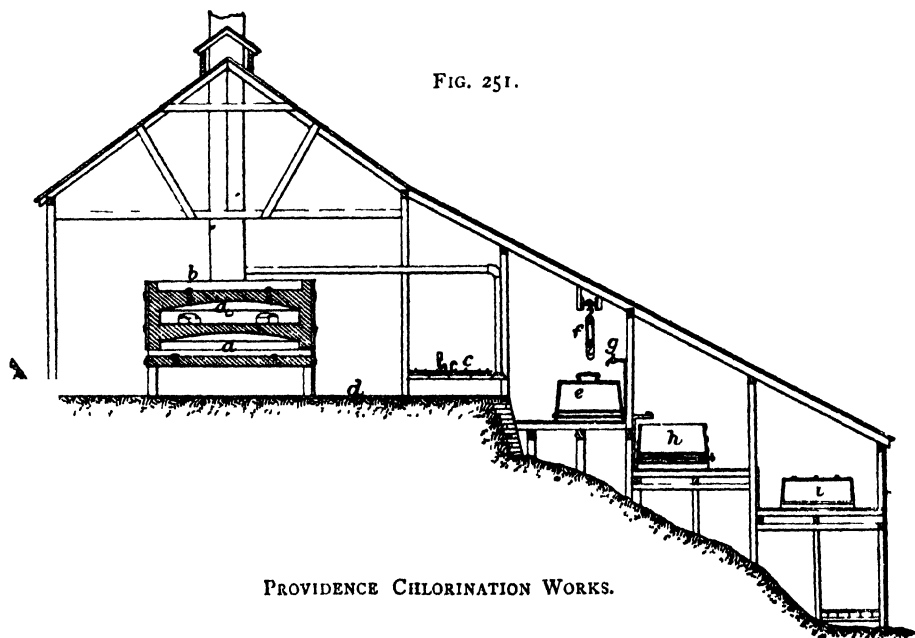
The reason why this process has not been more generally introduced is its inapplicability to all low grade ores, and especially to those which have a gangue which may be attacked in the course of the treatment. There is a loss of the coarser particles of gold, which have not had time to become attacked. It has therefore been proposed to treat the residues from the tanks in sluices over mercury riffles. A considerable quantity of gold might thus be saved, as the gold is bright and would amalgamate easily. The general loss in the whole process when the ore contains no silver to be extracted is less than 5 per cent. By using riffles it could probably be reduced to much less. By using leaching after chlorination most of the loss is obviated.

In selecting a site for chlorination works, the fall of ground, supply of water, and the direction of the prevailing wind must be taken into account. Ground must be selected which has a considerable fall, so that

the ore or tailings may arrive at the top of the roasting furnaces. This fall should allow, at least, of the use of an inclined reverberatory, and if possible of a terraced furnace, or of a mechanical furnace. If all the ground available was necessary to the leaching, elevators might be used for the ore. From the bottom of the furnace there should be a sufficient incline to allow the liquids to flow into the vats where they are to be used without pumping. The supply of clear water should be at least 40-50 gal. per hour, and there should be several days' supply of it on hand. The water should be carefully tested to ascertain that it will hold the gold in solution, as waters containing organic matter will not. The gases given off from the furnaces and vats are destructive to vegetation and machinery, and are very injurious to health. The works, therefore, must be situated so that the prevailing winds will not carry them in the direction of any mines, mills, or dwellings. The works are generally arranged in a long rectangle, one side of which is occupied by the roasting furnaces. The impregnation vats are in a straight line with the chlorine generators in the middle or to one side, generally in front of the impregnation vats. The precipitating vats are at one end of this line, so that the furnaces occupy one side of the works and the vats the other.

Figs. 251, 252, show the Providence Works. The 2 two-hearth

FIG. 251.

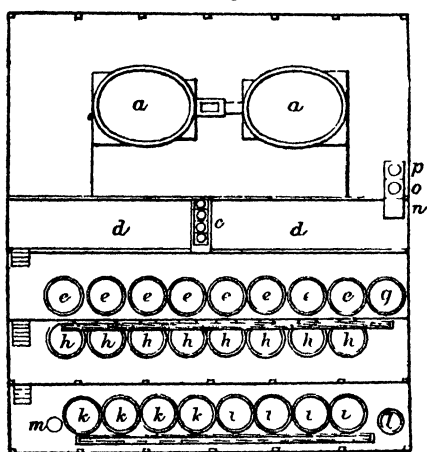


PROVIDENCE CHLORINATION WORKS.

roasting furnaces *a*, with a top *b* arranged for drying the ore, are on the highest level. The 4 chlorine gas generators *c* are on the same level

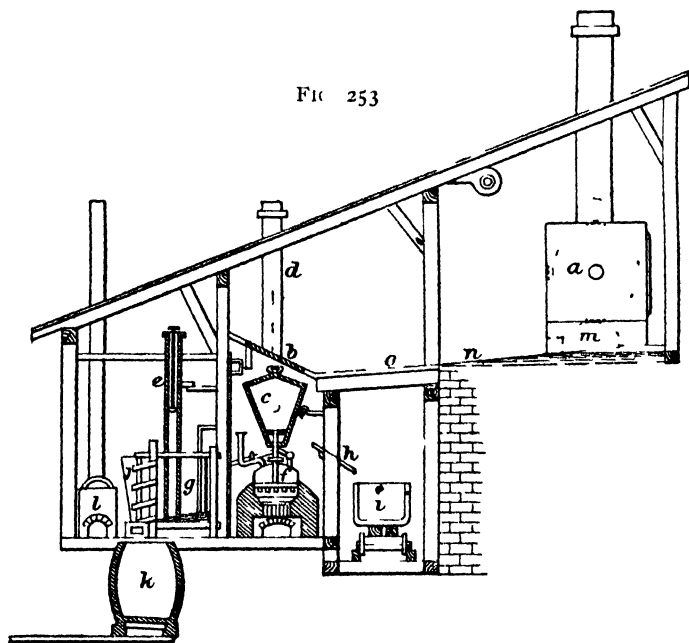
with the furnaces *a*, and in the centre of the brick cooling floor *d* in front of them, and divide this floor into two, so that there is one floor for each furnace. On the level below are 8 chlorination tanks *e* with an overhead railway *f* for the removal of the covers. On one end of the line is the vat *g* for the hyposulphite solution. On the level below, but contiguous to the chlorination tanks, are 8 silver-leaching tubs *h*, and on a still lower level 8 other vats, 4 *i* being used for precipitating the silver and 4 *k* for precipitation of the gold. At one end of the line of silver tubs is the well *l* for the collection of the hyposulphite solution to be used over again. Next to the end of the line of gold tubs is the tank through which the gold solutions run after precipitation to collect the floating gold. *n* is a furnace, *o* a water heater, and *p* a sand heater.

FIG. 252.



PROVIDENT WORKS

FIG. 253

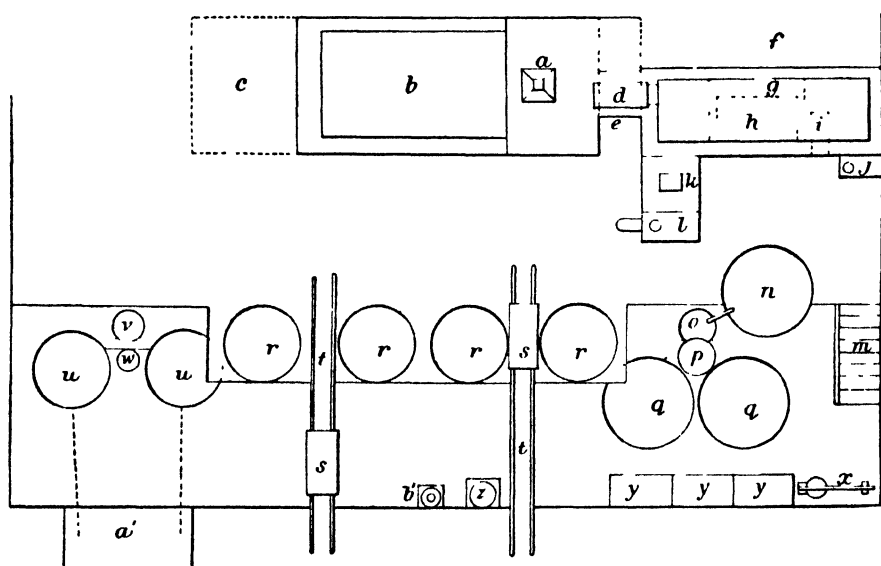


LINCOLN CHLORINATION WORKS.

Fig. 253 shows another works arranged in steps with the roasting furnace *a* at the highest level. The ore from the furnace is discharged

on to the sieve *b*, just above the impregnation vat *c*, which is suspended on gudgeons. The covers of the vats are raised by a pulley *d*, the rope of which is attached to rings *e*, on the side of the vat. There are 6 of these vats, 3 on each side of the chlorine gas generator *f*, which delivers the gas under the bottom of the tub. In front are the precipitation vats *g*, 2 for each set of 3 impregnation vats; the spent ore from the vat *c* is slid over a shute *h* into the spent ore *i*. The supernatant liquor is drawn off the precipitated gold at different levels by the taps *j*, and runs into a large vat *k*, which is filled with sawdust to collect the gold remaining in

FIG. 254.



PLAN OF UNION HILL CHLORINATION WORKS.

the liquor, and from here runs to waste. *l* is a crucible furnace; *m*, cub for roasted ore; *n*, brick cooling floor; *o*, wooden floor, 30 ft. by 6 ft.

Fig. 254 gives the plan of a works designed by Aaron for treating gold ores containing silver. The whole of the dry part of the process, including the roasting of the ores and fusion of the gold, is done on one side of the works; the leaching process is done on the other side and at a lower level. The ores are first chlorinated in the vats *r*, and the gold liquor is carried to the precipitation vats *u* at the opposite end of the line. It is then leached for silver in the same vats *r*, and the silver liquor is conveyed by launders to the silver precipitation vats *q*. Two tramways serve the tubs, the spent ore being discharged into the tail cars *s*.

The Pioneer reduction works are situated about $\frac{3}{4}$ mile south-west of

Nevada City. They were started in 1858, and have been in operation ever since, having in the interim treated 16,800 tons of concentrates. In 1888, 1089 tons were handled. The average value of the material treated has been \$80 per ton, of which 94 per cent. has been extracted. The 2 roasting furnaces are of the kind known as the long reverberatory—one being 50 and the other 60 ft. in length. The first hearth in each is 16 ft. long by 12 ft. wide, the other hearths having a width of 10 ft. These furnaces have a joint capacity of 5 tons in 24 hours, with a consumption of $\frac{3}{4}$ cord of wood per ton. The fuel is mostly pine and cedar, costing an average of \$4 40c. (18s. 9d.) per cord. At present the works employ 5 men at \$2 50c. (10s. 5d.) per day each; when running to the full capacity, 9 men are required. The furnaces are in a separate building, the roasted ore being carried to the leaching house, 500 ft. distant, by means of an iron car and tramway. There are 9 gassing tubs, each 5 ft. 6 in. diameter by 4 ft. 4 in. deep in the clear, with an average capacity of 3500 lb. of ore each. There is also a set of 9 leaching vats for silver, to which ore containing that metal is transferred after the gold leaching, and is then treated by the Kiss process for the silver. These silver vats are 3 × 4 × 5 ft. in size. The 4 precipitating vats for gold are 30 in. deep and 8 ft. in diameter; there are two similar precipitating tubs for silver. The concentrates are usually bought, but little custom work being done. The charge for custom work, in small lots, is \$18 (3l. 15s.) per ton. The purchasing price is usually 92 per cent. of the gold assaying value, no allowance being made for the silver unless the value in that metal exceeds \$10 (2l.) per ton, when 60 per cent. is given. The charge for working is deducted, so that, in fact, the transaction amounts to exactly the same thing as custom work, at a fixed price with a guaranteed percentage extraction. The percentage is calculated on the assay of the raw material, hence any loss sustained in the roasting, etc., falls on the purchaser. A charge is drawn from the first hearth of a furnace every seventh hour, which gives the ore over 24 hours' roasting altogether. The roasted ore is in chlorine 36 hours before the leaching begins. The gold is precipitated in the usual way by iron sulphate.

The chlorination works of the Merrifield Mine are situated on Deer Creek, about 1 mile west of Nevada City. The lessee buys concentrated sulphurets from the various mills, paying 85-90 per cent. of the result of sample assay. He roasts the ore before taking the sample for assay, in order to protect himself in case there be a loss of gold in the roasting, which he thinks always occurs to a greater or less extent. His charge for custom work is \$18 (3l. 15s.) per ton. These works consist of a reverberatory roasting furnace 70 ft. in length, by 10 ft. wide inside, and having a roasting capacity for 3 tons of concentrates per 24 hours, with a consumption of $\frac{3}{4}$ cord of wood to the ton. The works contain 2

chlorine generators, 4 chlorinating vats, 3 precipitating tubs for gold, 3 for silver, and other usual implements and apparatus. The dimensions of the vats are: diameter 6 ft. 8 in., depth 2 ft. 6 in. inside, with a capacity for 3 tons of ore. The precipitating vats have a diameter of 6 ft., and a depth of 3 ft. The vats are coated with a mixture of asphalt and coal tar. Red fireproof paint is used for the inside of the vat covers, this protecting the wood from the action of the chlorine perfectly. The filters are simply perforated boards covered with burlap. After the gold has been leached out, the ore, if it contains silver, is transferred to the silver leaching tubs, where it is leached with calcium hyposulphite for the silver. The latter, after being thus extracted, is precipitated as silver sulphide in the precipitating tubs, by stirring in a solution of calcium polysulphide. On the sides of the silver precipitating tubs a thick black crystalline crust has formed, which contains a large proportion of silver, it being probably gypsum and silver sulphide. The chlorine generators are of the pattern formerly used by Prof. Crosby, at Nevada City, being without stirrers. They stand in a slightly inclined position, so that the bottoms slope downward from the point of ingress of the acid. A water bath is used for heating the generators, it being preferable to a hot plate, as it prevents too high a degree of heat, which causes the solid in the generator to form a hard crust, troublesome to remove. The tops of the generators are of the flat style, luted in a flange. They very seldom require to be raised, as, to avoid the danger of too high heat, the exhausted charge is readily removed by means of a stream of water, through the spout in the side, which passes through the wall of the iron pan forming the water bath. For the eduction of the chlorine there is a short, wide standpipe in the cover, in the upper end of which is fitted a plug of lead carrying the eduction pipe, also of lead. Another similar arrangement carries the S tube by which acid is introduced, this being made wide enough to serve also, on removing the plug, for the introduction of the charge of manganese and salt. Sulphuric acid is bought in iron tanks, which effects a saving in freight and breakage, as compared with carboys. The ore is exposed to the action of chlorine during 48 hours. The gold is precipitated from the leach by solution of iron sulphate.

The Angels chlorination works have a capacity of 3 tons per day. The furnace is 80 ft. in length, 13 ft. in width, and 6 ft. high, outside measurement; the stack is 2 ft. by 2 ft. in the clear, and about 40 ft. high. The walls of the furnace are 1 ft. in thickness; the bridge wall is 13 in., and from the top to the grate bars is 14 in., and the drop from the bridge wall to the hearth is 12 in., the ash pit is 13 by 22 in. There are 9 doors on each side of the furnace. One ton of dried sulphurets is introduced into the furnace every 8 hours and moved forward successively to that portion of the furnace called the middle

oven, into which 20 lb. of salt are distributed and thoroughly mixed with the charge; the sulphurets are gradually brought forward to the hottest part of the furnace called the third oven, and after remaining there about 8 hours are drawn and spread on the cooling floor. There are 3 chlorination tanks, 9 ft. diameter at the bottom and $3\frac{1}{2}$ ft. deep. About 4 tons of the roasted charge, properly moistened, are screened on to the filter prepared in the bottom of the tank. Three gas generators are used for one tank, and each requires 32 lb. manganese, 36 lb. salt, and about 6 gal. sulphuric acid 66° B. "Gasing" is continued for 6-8 hours, and the charge is allowed to stand in gas 2 days. Water is then turned on, and allowed to remain in the tank about $\frac{1}{2}$ hour; it is then run into the base tub, to which 1 gal. sulphuric acid is added; from the base tub the solution is siphoned into 2 precipitating tanks, and proto-sulphate of iron is added until no more precipitate is formed. The solution is allowed to stand 2-3 days in order to completely settle; the clear solution is then siphoned off into a sluice box, which conveys it to outside tubs for further treatment if necessary. About $1\frac{1}{4}$ cords of pine wood, costing \$5 (14s) per cord, are consumed in roasting 3 tons of sulphurets; 3 men are employed on the furnace at \$2.50 (10s. 5d.) per day, and 1 chlorinator at \$3.50 (14s. 7d.) 1 floorman at \$2.50 (10s. 5d.) an assayer, and a superintendent.

The Phoenix Reduction Works, Drytown, and Amador Reduction Works, Sutter Creek, are owned and operated by Voorheis & Barney. The works at Sutter Creek have been in successful operation for a period of 15 years, and have a capacity for treating $4\frac{1}{2}$ tons of sulphurets a day. The Drytown works have been in operation for 12 years, and have a capacity for treating 3 tons a day. The size of the roasting furnace at Sutter Creek is 14 by 80 ft. on the outside. The works contain 6 large tanks, capable of leaching $4\frac{1}{2}$ -6 tons each a day, and 6 settling and 6 precipitating tanks, of suitable sizes. By permission of the owners, the following figures are taken from the books of the company, giving the actual cost of treating by chlorination, for a period of 6 years just past, at the Sutter Creek works, during which time 5136 tons of sulphurets were worked.

All assays are made immediately on delivery of material, and the following percentages are paid as soon as the assay value of the material is known, and before the sulphurets have been subject to treatment:

For sulphurets assaying \$50 and over per ton, 90 per cent., less \$20 charges for working.

"	"	100	"	"	92	"	"	20	"	"
"	"	200	"	"	93	"	"	20	"	"
"	"	300	"	"	94	"	"	20	"	"
"	"	400	"	"	95	"	"	20	"	"
"	"	500	"	"	96	"	"	20	"	"

If the sulphurets contain silver in appreciable quantity, payment is made for the silver, according to special agreement.

The sulphurets are received at the works in sacks, usually containing the same amount. Each sack is weighed and sampled, and the sample is immediately sealed in bottles to prevent a loss of moisture, and a duplicate is sent to the seller. Special arrangements are sometimes made whereby the sellers do not pay for delivery, owing to the grade and character of the sulphurets, quantity furnished, distance hauled, &c. These works, at Sutter Creek and Drytown, furnish the producers of sulphurets, in this and adjoining counties, an opportunity of realising an immediate cash value upon their material, and as quickly as they can on gold bullion sent to the mint. The works at Drytown cost \$12,000 (2400*l.*), and those at Sutter Creek \$15,000 (3000*l.*).

The following is a statement of the cost of labour and superintendence, and the amount and cost of materials consumed in connection with working 5136 tons of sulphurets :

EXPENSE	Total Amount		Pounds Per Ton.	Cost per Ton.			
	\$	c		\$	c	£	s. d.
Labour and Superintendent's salary, at \$200 per month	39,339	00	..	7	66	1	11 11
82 tons peroxide of manganese, at \$40 per ton ..	3,280	00	32	0	64	0	2 8
128 tons salt, at \$15 per ton	1,920	00	50	0	38	0	1 7
308,160 lb. sulphuric acid, 66°, at 3½ cents. per lb.	10,785	60	60	2	10	0	8 9
2568 cords of wood, at \$6 per cord	15,408	00	½ cord	3	00	0	12 6
General expenses for 6 years, including assaying, repairing, sundry supplies, hose, tools, &c., insurance, taxes, water, new vats, interest on capital invested, repairs on furnace and buildings	19,003	20	..	3	70	0	15 5
Total expense	89,735	80	..	17	48	3	12 10

To Aaron is due the discovery that there may be a heavy loss of gold by volatilisation during the roasting of auriferous pyrites under certain conditions. The loss in his case was due to the addition of about 2 per cent. of salt to the raw charge, on account of silver contained in the ore; and when this was discovered, the method adopted, which was successful in preventing any appreciable loss of gold, was to complete the dead roasting (as to iron), then cool the charge somewhat, add the salt, mix it with the ore as quickly as possible, and immediately withdraw the charge from the furnace, allowing it to remain in the hot pile for some hours before being spread to cool. It was found, however, that in this way the extraction of the silver was not so satisfactory as before; this was remedied without detriment to the gold, by the addition of 2-3 lb. finely ground salt per ton to the raw charge, the

remainder being added at the finishing. As Aaron did not ascertain the percentage of the silver which was soluble before the last addition of salt, this latter may have been superfluous. In some muffle experiments by Prof. Christy, more gold was lost when the salt was added after an oxidising roasting than when it was mixed with the raw ore; but in Christy's experiments the roasting was continued after the addition of salt, while in Aaron's it was not. Aaron's method is now in general use throughout California where the Plattner process is practised. Stansfield, of Nevada City, declares that there is some loss of gold in roasting without salt. In one instance it amounted to 25 per cent. of the value, but it is not stated whether the ore contained tellurium or selenium.

Christy has proved that gold is volatile in the presence of chlorine, at all temperatures from that of boiling water to a white heat, and that the amount of the loss depends upon time, heat, surface exposure, and quantity of chlorine. Gold thus volatilised, at least at the temperature of roasting, cannot be recovered by simple condensation of the fumes; it is necessary to employ some agent which can decompose the volatile gold chloride. It appears that sulphur dioxide in the presence of water does this, just as it decomposes the ordinary gold trichloride dissolved in water, precipitating the gold in the metallic state. In a long furnace, with several hearths, the salt is added on the finishing hearth; at the same time the ore on the next hearth is burning and giving off fumes of sulphur dioxide; hence it is only necessary to draw the smoke from the furnace through a sufficiently high coke tower, the coke being constantly wetted by a small stream of water, when the gold fume will be decomposed, and the metal be precipitated on and among the coke, or remain in part suspended in the water, which would be filtered before running to waste. The same water may be elevated and repassed until it becomes too acid. Charcoal may be used instead of coke, and Aaron suggests that probably quartz would also answer. The coke or charcoal has to be burned in order to recover the gold, but quartz would only require washing. It would be necessary to force the draught by suction, a fan or blower, made of wood, and painted with asphaltum and coal tar, a small wheel being used to operate it where water power is cheap. A steam jet would probably answer, and the steam, condensing partly in the tower, would do no harm at least. But it is doubtful if such an arrangement would be profitable, so long as the roasting is properly conducted. A loss of gold may also occur through too rapid roasting, the heat being too high, and the fine particles of metal being carried away by the violent evolution of the gases of combustion. Prof. Christy has shown that iron pyrites can decompose the volatile gold chloride, thereby becoming enriched.

Aaron does not approve of a storied furnace, one hearth above another. Three hearths on this plan would be very inconvenient, and when there are but two, the upper, on which the new ore is laid, is liable to be heated too much from below, where the heat can do little good for want of air to enable the sulphur to burn. What the ore needs, at first, is just enough heat to cause the sulphur to burn, and plenty of air; then stirring only when the surface glow begins to fail, the heat being gradually increased as requisite to continue the combustion, until on the finishing hearth, it is raised to that necessary for the complete decomposition of all iron sulphate previously formed and not yet decomposed. No recent important advance has been made in the means of roasting auriferous pyrites in California. The reverberatory furnace with 3 or 4 hearths still holds its ground, though rotating cylinders of the Brückner type have been successfully applied, and no doubt inclined cylinders of the White type may be equally so, if intelligently used, and with suitable modifications to meet the requirements of this kind of work.

A notable loss of gold by volatilisation was proved also by Stetefeldt, in 1885, when examining the gold ores of Las Minas, in the State of Vera Cruz, Mexico, with a view of finding a cheap and efficient method for extracting the gold. The following minerals could be seen in these ores: magnetite, iron pyrites, copper pyrites, quartz, and garnet. Neither galena nor zinc-blende was visible, but the ore contained a trace of lead, and some samples from the Muertos mine, 1·19 per cent. of zinc. Antimony and arsenic were not present. Native gold only appeared—mainly in extremely fine particles—after pulverising and washing. Magnetite was the most prominent mineral, varying between 43 and 67 per cent. Determinations of copper and sulphur showed that $3\frac{1}{2}$ –7 per cent. copper pyrites, and 3–22 per cent. iron pyrites were present. The gold-value of the ores varied between 0·3 oz. and 0·8 oz. per ton. All contained some silver, partly alloyed with the native gold, partly neutralised; but the silver-value was so low that it had to be entirely neglected in the metallurgical treatment.

For the experiments, the ore was crushed wet by stamps with No. 40 screen on the mortar, and allowed to settle in a tank. In the sluice-boxes, between the battery and the settling-tank, the coarser particles of the native gold were caught, to avoid errors of sampling, produced by coarse gold, in subsequent metallurgical experiments. The roasting was done in a small reverberatory furnace.

The results attained showed :—(a) That mere oxidising-roasting left a large percentage of magnetite intact, and that continuation of the process was of very little benefit after the pyrites had been roasted dead. Some aluminium sulphate, resulting from decomposition of the garnet, remained undecomposed, but that would not interfere with any subsequent

treatment. (b) Direct chloridising-roasting removed the magnetite almost completely and very rapidly. (c) If the chloridising-roasting was continued beyond a certain time, magnetite was re-formed, cuprous chloride becoming oxidised by withdrawing oxygen from ferric oxide. (d) An enormous loss of gold by volatilisation, which, no doubt, took place with that of the copper chlorides. The loss increased with the quantity of these chlorides formed and then volatilised.

But that copper chloride is not an essential element in producing this loss is shown in the following experiments by Butters, with a gold-ore entirely free from copper. The gangue was very hard whitish quartz, intimately mixed with about 7 per cent. calcite. It contained 5.55 oz. silver, and 0.65 oz. gold per ton. In ordinary specimens neither silver mineral nor gold was visible to the naked eye. After concentrating a quantity of the ore, pulverised so as to pass a No. 40 wire-screen, and examining the concentrates under a powerful microscope, a dark steel-gray silver mineral was revealed, covered with minute crystals of gold. In several samples examined, all the gold appeared to be attached to the silver mineral, and none was seen by itself. Besides, there was some iron pyrites. In subjecting the ore to oxidising-roasting, it assumed a pinkish colour, and lost about 3 per cent. in weight, corresponding with the amount of carbonic acid driven out from the calcite. A loss in silver from 2 to 9 per cent. took place, but none in gold. Very different were the results in chloridising-roasting. The experiments were conducted in the muffle, at a cherry-red heat. In roasting several samples for 1 hour with 5 per cent. salt, 70–80 per cent. of the silver and 68–85 per cent. of the gold were lost. The chloridised ore assumed a dead-white colour. After leaching it with water, the solution gave only a faint chlorine-reaction, the salt having been completely volatilised, and with it the larger portion of the precious metals. Roasting with 10 per cent. salt did not increase the loss in silver and gold. The ores were of an uncommon and peculiar character and contained the gold in exceedingly fine particles.

The loss in gold may be reduced to a minimum by instantaneous roasting in a Stetefeldt furnace; and if it were possible to condense the metallic fumes escaping in roasting the Las Minas ores, this would be a simple method to extract not only the gold, but also a large percentage of the copper. From Stetefeldt's experiments, it appears that the copper chlorides are most effective agents to volatilise the gold. These chlorides are also, it appears, much easier to condense than other metallic chlorides. Walker's experiments on electrical collection of metallic fumes and dust may point to a successful solution of this problem.

Plattner's chlorination process has also been introduced, with modifications, by the United Pyrites Co., into Sandhurst, Victoria. An account

of their *modus operandi* has been most obligingly communicated to the author by C. W. Chapman, the Co.'s manager, which may best be reproduced in his own words.

"The operations to which concentrated pyrites is subjected are the following :—(1) Oxidation and sometimes partial chlorination by roasting in a revolving furnace ; (2) impregnation of the roasted pyrites in vats ; (3) leaching with cold water, and filtration of the solution ; (4) precipitation of the gold by sulphate of iron ; (5) smelting the auriferous residuum. The finer the concentrates, the quicker and easier will the process be performed.

"By pyrites, is here meant the sulphides and arsenides of iron, copper, zinc, and lead. The presence of galena, blende, or copper-pyrites does not interfere with chlorination ; but when they are present, the roasting must be protracted, and more carefully performed. Gold alloyed with 10 to 20 per cent of silver is more easily obtained by this process than when in a state of comparative purity. Coarse gold does not admit of chlorination. Pan-tailings allow a perfect extraction of the gold by this process. The presence of lime or magnesia makes chlorination of the roasted pyrites troublesome ; the use of salt in roasting partially removes this difficulty. If lime, baryta, &c., be present, hydrogen sulphide may be used to precipitate the gold, and then the earthy metals will remain in solution.

"The roasting is the most important part of the whole process, and that on which the success of the subsequent operations depends. The sooner after concentration that the pyrites is roasted the better, so that no crusts or lumps may be formed by partial decomposition ; any hard lumps must be sifted out, and subjected to some disintegrating process. Not more than 1 per cent. of the sulphur shown by analysis should remain after roasting in an unoxidised state.

"The construction of the furnace has but little influence on the chemical results of roasting ; the best construction is that where the flame of the fuel comes into direct contact with the pyrites. The chief thing is to keep the cost down, and this is obtained by revolving the furnace, and so obtaining a mechanical stirring.

"In all furnaces, at the beginning of roasting at a low temperature, the sulphur of the pyrites is set free, combining with the oxygen of the air to form volatile sulphurous acid gas, which is well known by its odour. The metals, by losing a part of their sulphur, are converted into oxides, and sulphates. Sulphate of iron is the precipitant of gold from the chloride ; its presence is therefore objectionable, and it is necessary to increase the heat by degrees in order to decompose the sulphates, and to form oxides. Similar is the behaviour of arsenical pyrites ; arsenious acid and sulphurous acid escape under the influence of heat and oxygen,

\$2.50 (10s.), which makes the total cost about \$9 (36s.) per ton of actual expense on each ton of 30-oz. ore.

Plattner's Chlorination Process.—This was the origin and foundation of the crowd of recent chlorination methods. Its principle is to attack the gold in the ore by chlorine, in order to produce a soluble chloride of gold which can afterwards be dissolved out by water. The chlorine is brought into contact with the ore in a gaseous form. This process was introduced into America by Deetken in 1858, since which time it has been largely used, with some local modifications, for treating the concentrated sulphurets from the gold mills. These consist mainly of iron pyrites, with smaller proportions of copper pyrites, arsenical pyrites, and galena, and more rarely tellurium; and range in value from \$40 to \$300 (8*l.*–60*l.*) a ton.

The process, as developed in California and Colorado, has been minutely described by Prof. Egleston, from whom the following account is borrowed.

To ensure the best results there must be no material present which will be attacked by chlorine except gold; this must be in a metallic state, either as it is mined or after its combinations have been destroyed by roasting. The chlorine must be purified from everything that would be likely to cause undesirable reactions. The concentrates, after they are delivered to the mill, must be kept wet until they are ready to go to the drying floors, just previous to their introduction into the roasting furnace. If they are allowed to dry, decomposition of the pyrites commences, forming lumps, which must be crushed before roasting. Oxidation of the iron may cover the gold with a coating which is not readily removed, and from such material the sulphur is not all driven off. Some of the precious metal in such cases will not be attacked, and will consequently be lost in the tails. It is not generally advantageous to treat ores poorer than \$25 (5*l.*) per ton, but ores as low as \$15 (3*l.*), and sometimes lower, are occasionally mixed with richer ores. The sulphurets treated contain extremely variable amounts of gold. Only ores with a completely insoluble gangue can be treated. Of these, all are not equally well suited to this process. If the gold is in coarse particles, the action of the chlorine is so slow that the process becomes impracticable. All the ores are sulphurets. The average composition is given below (p. 610).

The process consists of four different phases:

1. Roasting the ore.
2. Impregnating the roasted ore with chlorine.
3. Leaching the impregnated ore.
4. Precipitating and fusing the gold.

(1) Roasting.—The object of this is to convert all the metals, except gold, into oxides, so that they will not be attacked by the chlorine, leaving

ANALYSES OF SULPHURETS.

	Fureka and Idaho Mines, Grass Valley.	Washington Mine, Mariposa County.	Black Bear Mine, Klamath County.
Copper	0·85	0·00	0·00
Lead	0·78	1·50	0·00
Gold	0·02743	0·00914	0·0137
Silver	0·0068	0·0035	0·003
Zinc	0 00	1·34	0·00
Iron	40·65	30·85	42·05
Arsenic	trace	0·00	21·25
Sulphur	32·80	31·33	25·10
Silica	12·64	33·30	10·35
Alumina	0·10	0·00	0·85
Magnesia	3·50	0·00	0·00
Oxygen and loss by difference ..	8·65	1·67	0·38
	100·00	100·00	100·00

only the gold to be acted on. The impurities most to be avoided are sulphur, antimony, and arsenic, (causing the precipitation of the gold), and lead, lime, and magnesia (likely to be attacked by the chlorine). It is also necessary during the roasting to avoid the production of any soluble chloride of iron or any other soluble metallic salt, as such will act on the oxide of iron, and also precipitate the gold once dissolved, and leave it in the tails, in the ore tanks, after lixiviation. Rocks containing lime and magnesia absorb more or less chlorine. To counteract this a small amount of salt is carefully added towards the close of the roasting, after the ore has been completely oxidised. When the ore is free from impurities, and in contact with the air and salt, sulphurous and then sulphuric acid are formed in the furnace; these acting on the soda set the chlorine free, and this acts upon the gold. When lead is present, the salt acts on the sulphate of lead, producing sulphate of soda, and a more or less soluble chloride, and oxy-chloride of lead. If the salt were not added, these chlorides and oxy-chlorides would be formed at the expense of the chlorine.

The extraction of gold from gold-bearing silver ores is more unsatisfactory according to the length of time during which the ore has been roasted, and is worse the higher the heat. Loss of gold is occasioned by the elevated temperature necessary to get a high chlorination for the silver. Chloride of gold is in itself volatile, and the loss may be 40–90 per cent. of the gold contained in the ore. The presence of copper chlorides brings it up to a maximum. When the copper is thoroughly oxidised before adding the salt, so that the amount of copper chlorides is reduced to a minimum, the loss is very much less. Roasting is usually effected in some kind of reverberatory furnace built of common brick. Mechanical furnaces are not suited to works of limited capacity. Four kinds of reverberatory furnace are used, varying only in the disposition

of the hearths. The hearth may be single and short ; a series of hearths may be placed one over the other ; the hearths may be single and long, and in the same plane ; or they may be arranged in steps. The first variety is wasteful of fuel, and there is always danger that the ore will be sintered in it. The second, though economical in fuel, is inconvenient ; the doors should be on opposite sides of the furnace and at such a level that they can always be reached. The third and fourth varieties are most economical. The fourth is a little more expensive to build, but there is a great gain in keeping the ore in the different stages of roasting on different hearths, and then dropping it from one hearth to the other ; but the chimneys must be so arranged that no fresh or unfinished ore can drop on to the hearths below. Both these furnaces are very advantageous in economy of fuel and labour.

At Sutcr Creek a *fortschauelfellungssofen* is used ; the hearth is 12 ft. wide and 80 ft. long, inclined from the flue towards the fireplace without any break. It has 12 doors in the side for working the charge, and is capable of working 3 charges of 2400–2600 lb. in 24 hours. The distribution of the charges on this hearth is as if the furnace contained 3 charges, which are kept separate with the very greatest care. The hearth nearest the flue, where the ore is quite cool, is used for drying. The middle hearth is used for burning off the greater part of the sulphur, and here the ore is spread out very thin and is constantly rabbled. The last hearth nearest the fireplace is at as high a temperature as it is possible to raise the ore, without sintering it, and here a dead roast is effected. If the temperature should be so high that any of the ore became sintered it would have to be re-crushed and re-roasted.

The smaller the charge treated the better the work will be done, so that it is usual to limit the charge on each hearth to 2800–4000 lb. It is also essential that in moving the charges from one hearth to another they do not become mixed, as that lengthens the working considerably. A very low heat is required on the upper hearth, especially if the ore contains lead. The colour of the heat should be a dark red when the ore is stirred, but black on the surface in the daytime ; at night it shows dull red. Constant rabbling on this hearth is not necessary, but the ore should be moved from time to time when the upper surface becomes dull ; towards the end only should the temperature be high. Tests can then be made to ascertain whether all the iron and copper sulphates have been decomposed ; but generally the men know by experience when the ore is finished. If any amount of copper is present, a high heat is required to decompose the copper sulphate. In some works the finished charge is panned to get the colour of pure gold, and to see whether there are any undecomposed pyrites present ; but this is not usual.

At the Lincoln Works are 3 hearths arranged in steps, each 7 ft.

below the one above. The discharge for the ore is through a flue the width of the hearth. Each hearth is 16 ft. long and 15 ft. wide, and is of the same width throughout. The upper hearth has 1 door only on the side and 1 at the end; the middle has 2 on the side and 1 at the end; and the lower has 2 on the side only. The drop flue is 12 ft. wide and 12 in. long. In it dampers are placed to regulate the draught, and a door on the upper level for the greater convenience of drawing down the charge. The grate is 8 ft. wide and 16 in. long, and only 4 in. below the hearth line. The height of the middle is 2 ft. in the centre, and 1 ft. at the sides. The fire bridge is only one brick thick. The top is 8 in. above the hearth, and 12-14 in. above the grate. On one side of this hearth is a door for discharging the roasted ore into a vault or cub beside the furnace. This cub is 4 ft. by 6 ft., and opens towards the cooling floor. Rails are placed over it for the ore cars to run on, so that the men can conveniently throw the ore into the car. The damp ore is placed on the roof of the furnace to dry, and is charged from the roof through cast-iron openings 8 in. square, which are closed with a piece of cast-iron and covered with ore. As the ore falls into the furnace dry, the dampers should be closed, and dust chambers should be connected with each furnace. The charge is $5\frac{1}{2}$ -6 tons; $2\frac{1}{2}$ -3 tons of dried ore are charged on the upper hearth, $1\frac{1}{2}$ -2 tons on the middle, and 1 ton on the lower, so that the furnace works $1-1\frac{1}{4}$ tons a day, varying with the quantity of lead or other fusible substances in the ore. Two men work 10 hours each, and 3 men 8 hours at each furnace. About 2 cords of wood costing \$5-5.50 (20-22s.) per cord, are used in twenty-four hours.

At the Eureka Works the furnace has 2 hearths at different levels connected by a flue 7 ft. 10 in. high. The upper hearth is 6 ft. wide, 39 ft. long, and has 6 working doors on the side. The lower hearth has the same width but is very much shorter. The furnace has a chimney 25 ft. wide and 28 in. square. The charge is introduced on the upper hearth, and is advanced little by little towards the lower one; 9 tons on the upper hearth, and 1 ton on the lower. Two roasters work the ore. As the ore is finished on the lower hearth, a charge of 1 ton is pushed down from above, and a further charge of 1 ton is added on the upper hearth, making 10 tons in the furnace at a time. As the men work 8 hour shifts, the capacity of the furnace is 3 tons in 24 hours. In front of the furnace is a cooling floor on which the roasted material is spread.

At the Union Hill Works are 2 furnaces which treat 2 tons in 24 hours. One of these is 12 ft. by 14 ft., and has 3 hearths; the other is 17 ft. by 14 ft., and has but 2. The former is only a little smaller than the other, costs no more to build, and does as much work; it is therefore preferred. The ore is rabbled 12 hours on each hearth of the

2-hearth furnace, and 8 hours in the 3-hearth. It generally takes fire within a few minutes after it falls into the furnace. One cord of wood is used in 24 hours in the summer and $1\frac{1}{2}$ in winter. In the works, Fig. 254, is one double-hearth furnace.

At the Providence Works, Fig. 251, are 2 elliptical furnaces with 2 hearths and a drying floor on top, built of common red brick. The hearths are 15 ft. by 12 ft. and are banded on the outside with bands of 4 in. by $1\frac{1}{2}$ in. flat iron. There seems to be no advantage in this shape. It is much more difficult to build than the ordinary furnace, and more difficult to keep in repair. The ore, which contains \$120 (20%) in gold and \$12 (48s.) in silver, is charged on the top of the furnace, all the lumps being carefully broken up, so as to make it as uniform as possible. It is there dried without being stirred. When quite dry, the upper hearth being freed by the discharge of its ore into the lower one, the charging hole is opened, and the ore is pushed down on to the upper hearth through openings made for the purpose. The usual charge is 5 tons. On the upper hearth it is vigorously rabbled every 10–15 minutes. Two tons of ore are drawn on to the lower hearth every 12 hours, and the same quantity is added from the upper one, and as much fresh ore admitted from the drying floor. After the charge has been 4 hours on the lower hearth, the fire is allowed to go down a little, and 1 per cent. of salt is carefully rabbled into the ore. The fire is then raised to the regular temperature, keeping the ore at a bright red heat until the roasting is completed, or 4 hours for treating a charge of $1\frac{1}{2}$ tons, and 6 hours for one of 2 tons. On the lower hearth the ore is constantly stirred up to within $\frac{1}{2}$ hour of the time of discharging, when it is raked into a pile and left until it is discharged. One man tends each furnace, dividing his time between the middle and lower hearths. By discharging the ore in 1–2 hours after adding the salt, instead of in 4–6, it was found that more silver was saved but less gold. In the works, Fig. 253, a Bruckner's cylinder is used.

At the Deloro Mine, Ontario, Canada, the ore is dried in an inclined revolving cylinder, 20 ft. long by 36 in. diameter at the small end and 48 in. at the large. At the small end is a conical addition 2 ft. in length, so that the furnace is 22 ft. long. The fire passes through this cylinder to dry the ore, which drops mechanically into the roasting furnace. This is a revolving cylinder 30 ft. long by 5 ft. diameter on the outside lining, with $4\frac{1}{2}$ in. of firebrick, having 8 shelves running nearly through from end to end. These shelves are formed of key brick 9 in. long, so that they project $4\frac{1}{2}$ in. from the lining. The fumes are drawn out from the furnace by a fan, which also is used for the draught. The ore runs from the first cylinder through the pipe directly into the second cylinder, 20 ft. long by 48 ft. by 4 ft. diameter, with a $4\frac{1}{2}$ in. lining,

which has 6 shelves, where the roasting is completed. The ore which feeds furnace No. 1 is heated by the escaping gases of the second cylinder. The 2 roasting cylinders are jacketed with an air space, a covering of mineral wool, and then paper. In these furnaces 10 tons of concentrates have been roasted in 24 hours, so that the gold has been extracted up to 93-98 per cent. The roasted ore is left in a pile until the furnace has cooled somewhat, then spread out on the cooling floor in front of the furnace, and when quite cool sprinkled with water and made just damp enough to prevent any dust rising when shovelled. Any lumps separated from the mass are ground before being sent to the tubs.

There is a very great advantage in using a drop furnace, either like that at the Lincoln Works or that at Union Hill. Since the ore will be thoroughly scattered by falling from a height, it is more likely to be brought into contact with the chloridising gases discharged from the lower hearth. In this respect, the furnace at the Lincoln Works seems one of the best. The Stetefeldt is also in principle a superior furnace, and probably would work well with a large supply of ore for constant use.

The fuel used in all is spruce or cedar; seldom pine. The foundation walls are generally 2 ft. thick, made of stone up to a little below the fire line. The rest of the furnace is constructed of ordinary red brick, as the heat is not sufficient to make firebricks necessary. The walls are generally not more than $1-1\frac{1}{2}$ bricks thick; rarely 2, except the small sides of the drop flues, which are always 2 bricks thick. The walls of the fireplace are generally $1\frac{1}{2}$ bricks thick. The lower hearth, when built with common brick laid up in clay mortar, will last at least 4 years; the upper hearth at least 6 years, with occasional repairs. When the furnace after this period needs to be repaired, 5000-6000 bricks will be required for putting it in good condition again. When the ore is discharged into the cubs, the discharge hole is covered with an iron plate set in a recess made in the bottom of the hearth, on which a layer of some carbonaceous material is generally put, and then a layer of old ore to prevent caking or attachment of the ore to the iron cover. The spent ore above the carbonaceous material is generally moistened a little to make it pack. The ore is usually drawn from the second hearth upon the one below very slowly, so that it may be chloridised as far as possible, and from that to the third, and so on, the new charge being introduced on to the upper hearth immediately after the discharge from the lower ones. All this work is done by the men in charge of the furnace. The assistant is never allowed to discharge the ore from the second to the lowest hearth until the salt has been introduced into the first. The roaster takes charge of the ore on the first hearth, while his assistant pushes it down. It is at a dark-red heat, and consists of metallic

oxides, sulphates of alkaline earths, and a very small percentage of sulphides.

The heat of the furnace, previous to discharging the ore, has been brought up to a bright red. When the ore, on being very suddenly turned over, shows only a few sparks, salt is introduced and the chlorination commences. It is not usually put in until after 3-4 hours' continuous stirring upon the lower hearth. It is always desirable, whether lead is present or not, to add a little salt in the last roasting, as it reduces the amount of chlorine gas consumed, and gives a purer precipitate of gold, but it must be done cautiously and at a low heat. The quantity of salt varies from 5 to 100 lb., depending on the amount of silver, lime, magnesia, and lead present. If the quantity of foreign substance is very large, previous to its treatment with the chlorine, the roasted ore should be lixiviated to remove any soluble sulphates and chlorides which have been formed in them, and which might prevent the action of the gas. It is rare, however, that any large amount of such substances is present; when only a small proportion of impurities is contained in the ore, lixiviation is unnecessary, as the water used in damping the roasted ore appears to attack the alkaline salts sufficiently to permit the gold to be acted on. On an average, only about 60 per cent. of the salt used is decomposed. If the amount of silver contained in the gold is large, lixiviation is necessary to separate the silver after the soluble gold chlorides have been washed out.

Roasting must always commence at a very low temperature, to avoid the formation of salts, which are likely to be reduced in case other metals, especially lead, are present, and also to avoid the fusion or sintering of the ore; and not only must a low heat be maintained during the whole of the first stages, but all the ore must be vigorously stirred during that time. If, however, a low temperature were maintained to the end, all the sulphates which are formed would not be decomposed, and all the sulphur would not be driven off from the ore. If soluble sulphates were allowed to remain in the ore lixiviated, this would precipitate the gold in the tails. The roasting of the ore must therefore commence at a low temperature and finish at a high one, so that it may be roasted "dead," to decompose any sulphates that may be formed. During roasting, air is introduced through the working doors, and the ash-pit is kept entirely closed.

On the finishing hearth, the ore is raked first from one side to the other, and turned bottom side up with the spadelle, in order to renew the surfaces. It is finally made into piles, which are cut down little by little with the same object. When the ore will maintain nearly a vertical face, shows no bright specks, and is inclined to become black, which will generally be in 7-8 hours, it is ready for leaching. The largest amount

of gold in a condition to become soluble, and the least amount of chlorine gas consumed to put the gold in that condition is when the ore falling in the furnace has a slight violet colour. When the ore sparkles, and the sparks are numerous and bright, a large quantity of salt has to be used. A ton of roasted ore will occupy generally $24\frac{1}{2}$ cub. ft. This is derived from 2800 lb. of raw sulphurets, which occupy about $13\frac{3}{8}$ cub. ft. per ton. A ton of sulphurets will weigh 1450–1700 lb. after roasting, and will occupy about $17\frac{1}{2}$ cub. ft. It ought not to contain more than $1\frac{3}{8}$ per cent. of sulphur.

From the furnace the ore is drawn either into cubs or on a cooling-floor, which for a furnace roasting 3 tons a day must have at least 350 sq. ft. The surface of this floor is usually paved with brick, and enclosed with a rim of $\frac{1}{8}$ in. boiler plate 18 in. high. It is necessary that the ore should be entirely cold before being placed in the impregnating vats, as warm ore requires a greater quantity of gas, and less gold is likely to be extracted from it.

(2) Impregnation.—When the ore has become entirely cool, it is damped with 4–5 per cent. of water, so that it will pack in the hand but gradually fall to sand when the pressure of the hand is released. It is then either sifted directly into the impregnation vats through a 6-to-the-inch sieve, or after sifting is collected in wooden boxes holding about 50 lb. each and emptied gently into the vats. It would not do to shovel it in, as it would become too closely packed, and prevent the passage of the chlorine gas. The usual form of these tubs, which generally hold between 2 and 3 tons of roasted ore, is shown in Fig. 246. The same is used for both impregnation and leaching. They are made of the best sugar pine 2 in. thick, $7\frac{1}{2}$ –9 ft. diameter, and 3 ft. deep, and generally have a batter of 1 in. to the foot. This is done to prevent any space forming on the sides when the ore settles, and thus making points of easy passage for the chlorine gas. The vats are usually placed together in a row, each vat being inclined a little to the front so as to drain it completely. In front of the vats is a launder *a* to carry off the liquid running through into a reception vat. These vats are thoroughly painted inside and out with 3 coats, usually tar, in some cases asphalt varnish, in others a mixture of coal tar and asphalt melted together and put on hot. This is done to prevent action of the chlorine and leakage. They are repainted once a year. At the Providence Mine, the bottoms *b* of the tubs are made of 3 in. grooved planks, which are tongued. The staves are made of 2 in. planks with plain joints, but each joint is lapped on the outside with a batting $2\frac{1}{2}$ in. by $3\frac{3}{4}$ in. The tub is held together by 4 iron hoops *c*. With this construction, there is no danger that the iron hoops will be attacked by the chlorine. The diameter of the tubs at the top is 5 ft. 9 in., at the bottom it is 6 ft. 3 in. They hold $2\frac{1}{2}$ tons of

ore. The filter bottom *d* is made of 1-in. boards placed 3 in. above the bottom of tub and separated by cleats. The depth of the tub above the false bottom is 3 ft. 6 in. These boards are perforated with $\frac{1}{4}$ in. holes, and are covered with a double thickness of gunny cloth. A cover *e* is made of $1\frac{1}{2}$ in. stuff, tongued and grooved, and reinforced by 4 wide cleats *f* which are screwed on. A groove is cut on the circumference of this cover, into which the tub fits loosely. The cover is sometimes suspended by two iron supports on a rolling truck *g* above. These supports are attached to the cleats *f*. The vats contain usually 2–4 tons of roasted ore. In most works, these vats are placed upon the floor in a

FIG. 246

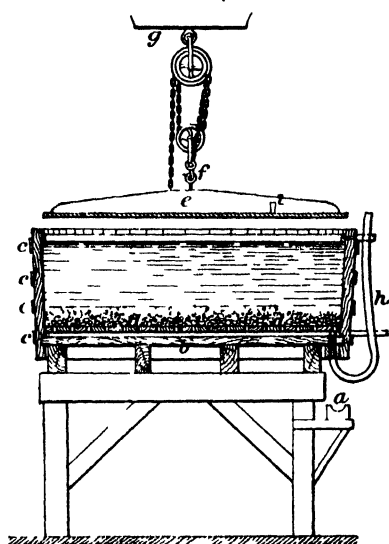
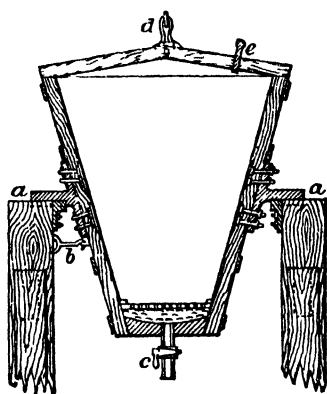


FIG. 247



IMPREGNATION AND LEACHING VATS.

line, and on one side of the roasting furnace. In some cases the vats are swung upon gudgeons, as at *a*, Fig. 247. In this case the tank is held in place by a hook *b*, and the gold chloride is discharged by a stop-cock *c* in the bottom of the vat. The number of these tanks varies with the quantity of ore to be treated. At the Lincoln Mine are 6, at Sutter Creek 5, at Providence 8. They have a false bottom placed at least $1\frac{1}{2}$ in. above the bottom of the tank, in order to allow the exit pipe, which is $1\frac{1}{2}$ in. diameter, to enter. This false bottom is pierced with $\frac{1}{4}$ in. holes, 10–12 in. apart. It is sometimes made by placing pieces of board filled with holes, 2 in. wide and 1 in. thick, 6–8 in. apart, over the bottom. These boards rest on cleats several inches apart, which are placed across the bottom of the vat, but do not touch the sides. As, in order to drain completely, the vats must be inclined towards the launder

in front, the cleats are made thicker on the front side, so that the false bottom is on a level. On this bottom a layer of quartz the size of a hen's egg is spread to a depth of 3-4 in. Upon this, smaller pieces half the size are laid $1\frac{1}{2}$ -2 in. in depth, and so decreasing in size until 6-8 in. from the bottom sand is placed. The filter is generally 8-10 in. deep. Over this, boards are sometimes placed, set in contact to form a surface to shovel upon when the leached ore is to be removed from the tank. Sometimes gunny sacking is placed over the filter. When the men are skilful they shovel down on to the filter without disturbing it, but this requires a great deal of practice. This filter remains permanently in the vat until it gets very dirty ; it is then removed to be washed and used over again.

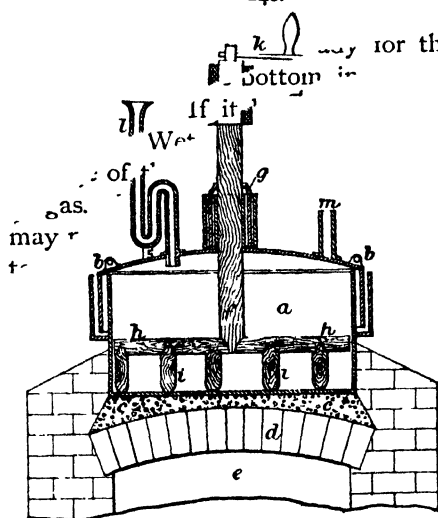
When the vats are ready for the charge, 15-20 buckets of dry ore are spread over the bottom, in order to take up any moisture which remain there. If it becomes too damp, a sufficient quantity of d_{be} is added. Wet ore over the false bottom would prevent the easy passage of the gas through it, and cause an increase in the consumption of gas. If it is too wet the whole quantity of gas used for the charge may not rise more than 2 in. above the bottom of the filter. When it is too dry the ore is not properly acted upon. If the bottom of the vat is not dried in this way the lower stratum of ore would take up the moisture and be too damp. The rest of the charge is moistened outside of the vat and treated in the same way. When it is of the proper dampness, it is raked together and charged with a shovel on to a brass wire sieve 4-8 to the inch, and falls on to the filter. This sieve has wooden sides and is 3-6 in. deep. It either runs upon rollers on two tracks placed at right angles to each other over the top of the vat, or is supported above it by ropes placed at each corner, so as to give it complete freedom of motion. In some cases it has handles on the sides, so as to avoid the labour of reaching over the front in order to get at all parts of the vat ; in others it has none, and the men are obliged to lean over the top to make the ore fall in the proper position. By moving the sieve backwards and forwards and from side to side, the roasted ore is distributed as evenly as possible, and is every few minutes raked evenly over the bottom by the man who throws the ore on the sieve. The object of the screening is to have the charge porous, to make it equally damp, uniform in size, and prevent nails and pieces of iron, which would precipitate the gold, from getting into the vat. In some works the ore is moistened and put into a pile, and is then sieved into wooden boxes, which hold about 50 lb. each, and their contents are carefully poured into the vat in such a way as to be sure that no packing takes place. The vat is filled in this way to within 2-4 in. of the top. It is necessary to leave this amount of space in order to be

sure that the whole of the chlorinated ore shall be covered with water when the leaching commences. If this were not so, the ore would be unequally moist in different parts, and it would be very difficult to get all the soluble gold out of it. When there is about 1 ft. of ore in the tank, the chlorine is turned on. It takes a man 2 hours to charge the vat. He must work rapidly and well, not to be caught by the gas, for by the time that the vat is completely filled the chlorine is near the top. On the top the ore is made highest on the sides and lowest in the centre, so as to be funnel shaped. It is then lightly pressed with the hand around the whole circumference of the vat in order to pack it and prevent the easy passage of the gas there. The gases have always a tendency to follow the sides of a vat if there is an equally easy outlet for them. When the vat is ready, a cover, which is made of flat boards strengthened on top with 6 wooden braces 2 in. wide and 3 in. high, is placed in position. This is coated on the inside with tar and suspended in many works by means of a chain and a Weston pulley, as in Fig. 246; in others there is simply a ring *d*, as in Fig. 247. Where there are a number of vats, they are placed in a row, and are served by a pulley, running on a rail *g* overhead. There are two ways of closing the vat; either a groove is made in the cover which fits over the edge of the vat, or the cover itself is made to fit the inside of the vat, as in Fig. 246, leaving a space between it and the sides of the tub, but resting upon a projection placed there for the purpose. This joint is usually caulked, first with rags which have been wetted, and then is made perfectly tight, either with a mixture of spent ore, bran and water, or with clay and sand of the proper consistency, or, as is generally the case, with tough dough, as this is the cheapest material that can be used for the purpose. It is always necessary to keep these joints moist during the process, by placing damp rags over them. The gas is introduced in some works by a single pipe *h*, Fig. 246, and *c*, Fig. 247, which enters under the false bottom; in others there is a pipe upon each side, which does not seem to present any special advantage. The pipe is made of lead and is introduced through the sides. It is enlarged from the inside, so as to make it perfectly tight, and then flanged. In some works a lead nipple, treated in the same way, is used to attach a rubber hose, and this seems to be the best disposition. A small hole in the cover is left for the escape of the air, and to observe by the ammonia test the moment when the chlorine reaches the top. This hole is then closed with a plug *i*, Fig. 246, *e*, Fig. 247, but the current is kept up until the charge in the gas generator has been completely exhausted. It is generally passed for 5-8 hours, and sometimes 12 hours, the time depending upon the size of the ore. When the gold is very fine, it will take less time than when coarse. To be economical, the operation must be conducted quickly.

Therefore, the gold should be as fine as possible. After the charge is thoroughly impregnated, the chlorine tube is removed, and the gas is allowed to remain in the ore for 24-40 hours. The best length of time, from entry of the gas to discharging of the tub, is 48-72 hours, depending on the fineness of the ore.

The chlorine necessary for a 3-ton charge of refractory ore is made in a covered generator *a*, Fig. 248, built of very heavy sheet or cast lead, or of iron lined with lead, the joints of which have been "burned" together, not soldered. This is 20-24 in. diameter and 11-12 in. deep. The cover is made gas-tight by a water joint. It is raised when necessary by rings *b* on the side. The bottom is made of lead, 16 lb. to

FIG. 248.



CHLORINE GENERATOR.

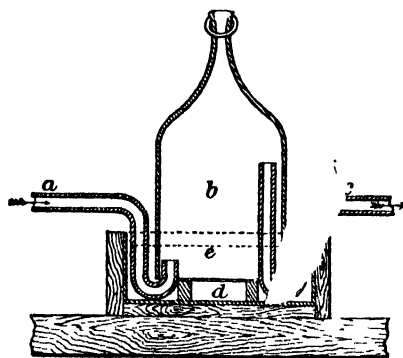
for 3 tons of ore consists of 20-24 lb. salt, 15-20 lb. manganese binoxide at 70 per cent., 30 lb. sulphuric acid at 66° B., and 18 lb. water. When necessary, the acid is carried to 45 lb. and the water to 25 lb. The water is not introduced until the gas is wanted. The cover is generally removed to put in the solids and water, but sometimes there is an opening on the top for the introduction of the charge without lifting the cover, and one in the bottom for removing the spent material. Both openings are closed with wooden plugs covered with rags thoroughly saturated with grease to prevent the action of the chlorine. The acid is introduced through a siphon *l*; $\frac{1}{2}$ lb. water is added to every pound of acid. This charge will be $3\frac{1}{2}$ -4 in. in depth. The acid is added in pitchers of about $\frac{1}{2}$ gal. at a time. In some works the acid is placed in a reservoir, whose stop-cock is directly over the siphon, and

the acid is made to fall drop by drop. Such a charge will generally run 10 hours. When ready to work, rubber tubing, which has been thoroughly greased on the inside, is placed upon the exit pipe *m* in the cover, which is $\frac{3}{8}$ in. calibre, and is attached to the entry tube *a* of the wash bottle *b*, Fig. 249, and the exit tube *c*, to which another hose is attached, which connects it with the leaching vat. The wash bottle generally is a large-sized acid bottle or an old carboy, from which the bottom is removed. It is supported in a wooden box *e* lined with lead and filled with water. The inlet pipes are held in place by a wooden support *d* in the centre of the box. The water in the box stands $\frac{1}{2}$ in. above the inlet pipe. This apparatus is simple, but requires that the pipes should be fixed so that there

would be no danger of their turning over, which they might do if not thoroughly supported. Sometimes a special washing apparatus is made. The sulphuric acid is introduced through the funnel, which is $\frac{3}{8}$ in. calibre, in small quantities at a time. It is absolutely necessary that the chlorine gas should be washed entirely free from hydrochloric acid, since, if any sulphides were present, they would be attacked, and the sulphuretted hydrogen would precipitate the gold in the impregnation vat. One stick of wood 6-7 in. diameter, 4 ft. long,

is sufficient to generate all the heat required to make the gas. The gas is given off slowly, and only at about the rate that it can be absorbed in the ore, but it must be given off uniformly. As soon as it commences to decrease in volume, fresh acid is added and the stirrer is turned. Occasionally the stirrer gets clogged, in which case it must be raised several inches, which can be done without allowing the chlorine gas to escape, as the water-joint is quite long. There will generally be no difficulty in breaking up the charge when the stirrer is elevated a little. At Deloro, the chlorine is made from a charge of 40-50 lb. chloride of lime and 50-60 lb. sulphuric acid. It is forced into a cylinder under pressure at 40-50 lb. to the square inch, and falls 25-30 lb. when the chlorination has been made as perfect as desirable. The operation lasts about 2 hours. The chlorine generator is generally placed in the centre of the row of vats so as to be as near as possible to them all. If, when the cover of the impregnation vat is lifted previous to the introduction of the water, there is a strong odour of gas,

FIG. 249.



WASH BOTTLE.

the chlorine has probably attacked the ore thoroughly. In the arrangement as designed in Fig. 253 the generator is placed directly beneath the leaching vat. In Fig. 252 it is placed in the centre of the row of vats in front of them, in Fig. 254 it is placed a little to one side. There may be one or more of them according to circumstances.

In the Davis process 150 lb. roasted ore is placed in a cylinder with 125 gal. water and 50 lb. calcium chloride. The cylinder is then rotated long enough to ensure thorough mixture; $3\frac{1}{2}$ gal. sulphuric acid diluted in the ratio of 1 to 3 of water is then forced into the cylinder, which causes the evolution of chlorine gas. It is generally best to attain a pressure of 25–40 lb. per sq. inch in the interior of the cylinder. The operation takes $\frac{1}{2}$ – $1\frac{1}{2}$ hours; the gas is then drawn off, which takes $\frac{1}{2}$ hour. The cylinder is discharged into the receiving tank, which has a quartz filter, and the ore is washed with 75 gal. water. All this is pumped into the reservoir, from which it is discharged into the precipitating vats, and the spent ore is sent to be treated for silver if it contains any.

(3) Leaching the Impregnated Ore.—When the charge has remained sufficiently long in the vat, the cover is removed. A great excess of chlorine gas completely fills the space between the top of the ore and the cover, and if it was lifted at once there would be considerable escape. This gas may be absorbed by water previous to lifting the cover, but as the gas is retained in solution it makes it difficult to settle the gold in the precipitation tanks. Aspiration is frequently used to draw off the excess of gas and to collect it in a receiver, for use in a subsequent charge. The introduction of the water for leaching the gold chloride must be done with precaution. In some works, to be certain of its equal distribution, the cover is lifted and gunny sacking is placed over the ore, previous to introducing the water; in others the gunny sacking is placed over the ore before the cover is put on, previous to the introduction of the chlorine, and without lifting it; the water is introduced in a hole through it and falls on the gunny sacking and is distributed over the ore. A much better way, proposed by Aaron, is to attach permanently under the cover a coil of lead pipe filled with small holes. One end of the pipe comes through the cover so that the water is sprinkled all over the ore in fine jets. In this way the most gold chloride is dissolved in the least amount of water, so that the first solution which collects under the false bottom is very rich, while that above is comparatively poor. In any case the vat is filled with water until it is about 3 in. deep over the whole surface of the ore, and the tub is allowed to stand until all the gas which is not absorbed by the water has escaped, which is done in 30–40 minutes. The rubber tube *b*, Fig. 246, attached to the exit pipe below the false bottom, is turned down into the launder *a*, which is of timber coated

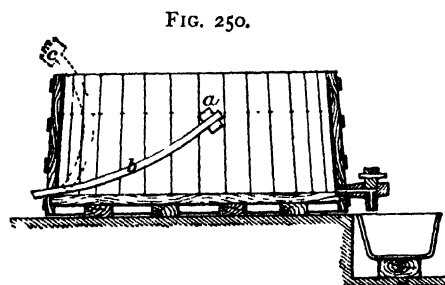
with tar. These launders are usually 4½–6 in. wide, and 3–4 in. deep. The solution is allowed to run out very slowly, and water is introduced at the top of the vat as fast as it flows from the bottom. The yellow solution, having a strong odour of chlorine, runs through a canvas filter into a vat 18 in. diameter and 2 ft. deep. In this vat, 2 in. from the bottom, a conical wooden funnel is placed, so that the rubber hose, 1 in. inside diameter, can be slipped over the small end. The object of the canvas bag is to catch any ore or sand escaping from the tanks. Through this tube the solution is carried to the precipitation tanks. This is continued until the solution running through shows no trace of gold with an iron sulphate solution. At the Lincoln Mine, no filter is used. The solution is run into tanks 5 ft. diameter to be settled, and is drawn off to be precipitated. The use of the filter facilitates the operation. In the settling vats, or in the precipitation vats when there are no settling vats, it is the practice in some works to add 32–40 lb. sulphuric acid at 66° B., as in such cases the gold precipitates very freely from an acid solution. It usually remains in the settling vats about 24 hours. The following analysis of the gold solution taken from a charge of 3 tons of ore yielding \$163 (32*l.* 12*s.*) to the ton, is given by Deetken. The sample was taken one minute after the opening of the discharge cock, and was allowed to settle 24 hours.

Sulphate of alumina	..	6·98 grains.	Chloride of iron	..	..	0·32 grains.
„	magnesia	.. 132·55 „	„	copper	..	8·95 „
„	lead	.. 4·16 „	„	sodium	..	536·00 „
„	soda	.. 426·62 „	Silica	..	..	0·24 „
Chloride of gold	..	10·44 „	Free chlorine, by difference	101·02	..	

When the ore contains lead, the soluble chloride is transformed into sulphate by the sulphates present, and gives the gold solution a milky appearance, but the greater part of this is left on the canvas filter, which must be frequently cleaned. If any of it were to remain in the liquor, it would be precipitated as sulphate by the vitriol, and would make the washing of the gold very difficult. In such case the liquor is drawn into a vat where the lead is precipitated by sulphuric acid. It is then decanted and treated as usual. The washing lasts 4–6 hours, and is continued until the last waters show no trace of gold. The ore is then allowed to drain. Sometimes all the waters from the last part of the washing and from draining, which are poor, are kept separate, to be used on the next vat of ore. An assay of the ore in the vat is now made. If it is still rich, which is very rare, it must be again chlorinated. In some works, when it assays \$15 (3*l.*) or over, the men are forced to do this at their own expense. If the assay shows that the ore has been properly treated, it is shovelled out into a car and carried to the dump heap. This is somewhat laborious. In some works the transfer of the

spent ore from one car to another is avoided by having a vat on wheels. It is rolled to the dump heap and there emptied. It requires some care to remove the ore from the top of the filter so as not to injure it. The labour is much reduced when the vat is suspended, as it can be turned over and the ore in it shovelled out. In other works, where the vat is suspended, when the vat is turned down its contents are washed out into a sluice by a hose. The spent ore then falls down a slide *h* into the spent ore car *i*, Fig. 253. The vat when empty is ready for the next charge. Every day one set of vats is charged with gas, a second remains in gas, and a third is filtered and emptied. When there are 3 or more impregnation vats, one set is filled every fourth day. Filtering and emptying are done every third day. When there are 6 or more, several are treated at a time.

(4) Precipitation and Fusion of the Gold.—The precipitating tanks, Fig. 250, vary from $5\frac{1}{2}$ to $6\frac{1}{2}$ ft. diameter at the top, and $6\frac{1}{2}$ –7 ft. on the bottom, and are 2–3 ft. high.



PRECIPITATING VAT.

They have no false bottom, and are generally made larger at the bottom than at the top to prevent the adherence of the gold to the sides. They are coated, like the solution vats, inside and out, or painted with paraffin paint, or are left without any paint of any kind, and are set entirely level to prevent the loss

of gold when decanting the waste liquors. They are made smooth on the inside to facilitate cleaning. The bottom must be made perfectly smooth and is shaped in such a way as to allow of the easiest removal of the precipitated gold. It is made either in Portland cement, or asphalt and cement heated together, and shaped as it cools. The precipitating solution is generally introduced first, so as to neutralise as much as possible the excess of chlorine present in the gold chloride solution, but care must be taken not to introduce too much of it. It is better to add a little afterwards rather than put in too much at the commencement. The gold solution is run into the tanks in such a way as to give it a rotary motion. This facilitates the mixture with the sulphate solution which is already in the tank, and causes the precipitated gold to collect. In some works the sulphate of iron is not introduced until the gold solution is all in the vat. In order to be certain that all the gold is precipitated, it is usual to save some of the strong gold solution which runs out when the first discharge is made, and to add some of the supernatant liquor from the precipitating vat to a small quantity of it.

whilst oxides, arseniates, and sulphates remain, and are further decomposed by increased heat. At the same time, all metallic iron derived from the abrasion of the stamps must be converted into an oxide.

"After the sulphur and arsenic have been driven off, provided salt has not been used, the gold remains free, and may be observed upon vanning a sample. When salt is used, according to Plattner, AuCl_3 is formed below red-heat; at 200°C . (392°F .), it changes to AuCl , and at a red-heat it is converted into metallic gold.

"Assertions are made from time to time that gold is lost in roasting; but proofs are wanting: experiments only show that gold in a very fine state of division is carried, to an inappreciable amount, into the flue by the draught.

"The furnaces used formerly by the United Pyrites Co. were reverberatory, 45 ft. long, 5 ft. 6 in. wide, with a sloping hearth, and port-holes for stirring the pyrites during the roasting process. These answered the purpose very well; but desiring a more economical roasting, a furnace of boiler-plate has been built, in two sections, each lined with brick, and revolving independently. The total length is 90 ft. One section is 3 ft. in diameter, and the other 4 ft., with a drop from one to the other.

"The time the pyrites occupies in the furnace depends on its composition, and this may be nearly enough ascertained by watching the behaviour of a sample roasting in the muffle furnace. The faster or slower the furnaces are revolved, the faster or slower will the pyrites be passed through. The roasted pyrites runs into a truck, and is then wheeled to the cooling-floor, where it is eventually moistened, and then trucked to the chlorination-vats.

"The roasted pyrites must be moistened, so as to lie in an open working condition in the vat, and because the chlorine gas will act more energetically upon the material in a moist condition than in the dry state. The moistening may be conveniently done with a hose, turning the mass over several times, so that no portion may remain dry; and the proper amount of moisture may be ascertained by compressing a handful to form a lump, which should retain its shape until handled. If too dry, more water may be added; and if too wet, more dry material.

"The vats are made of wood, to hold 1 to 5 tons, and are shallow in proportion to their diameter; they are coated inside with a mixture of pitch and tar, to prevent absorption of the gold solution: above the bottom is an empty space about 1 in. high, covered with a false bottom of perforated boards. The boards are supported by short pieces of wood, leaving sufficient space for the chlorine gas to pass. Over the false bottom a filter is made in the following manner:—1st layer, 1 to $1\frac{1}{2}$ in. quartz gravel; 2nd layer, $\frac{1}{2}$ in.; 3rd, $\frac{1}{4}$ in.; then coarse sand; and finally fine sand; making a filter about 5 in. thick; this filter remains always in

the vat, unless repairs have to be effected. Under the false bottom are three holes, two by which chlorine may be introduced, and the third for the discharge of the lixivium. After a charge has been removed, the filter retains a great deal of moisture, which is drawn up into the new charge; therefore some dry material is placed first upon it, to absorb this moisture.

“The moistened pyrites is sifted into the vat, that it may be evenly and loosely distributed, and to free it from any stones or crusts formed during the roasting. This is done by pushing the sieve to and fro upon two pieces of quartering over the vat.

“There are three 10-ft. vats, one 8-ft. and one 6-ft., all 3 ft. high. The covers are made gas-tight, and fit into a step, which is luted round with linseed-meal when the gas rises to the top. They are raised and lowered by a block and tackle.

“The chlorine gas is generated in an earthenware vessel of about 20 gal. capacity, like a large jar, only that the top, besides the central large opening, has two smaller ones, one containing a pipe, bent somewhat like the letter S, with the outer end widened for a funnel, through which the sulphuric acid is introduced; the other conveys the chlorine through a washing apparatus to the vat.

“The gas-generator stands on bricks in a wooden tub containing water heated by steam, and this tub stands on a truck so that it may be moved for cleaning the generator, or connecting with any one of the row of vats.

“When a vat has been charged within 6 in. of its top, two generators are put into position, and connected with the vat. Into each generator, for a 10-ft. vat, are put 16 lb. of manganese, 17 to 20 lb. of salt, 35 lb. of sulphuric acid, and 20 lb. of water; all but the sulphuric acid are introduced through the central opening, and lastly the sulphuric acid is run through the leaden pipe referred to before. The chlorine gas is not allowed to pass directly to the vat, but through a washing apparatus, made by half filling a wash-basin with warm water, and conveying the gas from the generator into it by a leaden pipe, the mouth of which is turned upwards, and placed $\frac{1}{2}$ in. under the water. Another pipe, turned the same way, but with its mouth above water, leads into the vat; over the mouths of both these pipes, and reaching down into the water, is a Winchester quart bottle with the bottom cut off. By this means the gas is washed free from hydrochloric acid. Twice a day is often enough to change the water, which is used warm because warm water absorbs less chlorine than cold.

“The object of this apparatus is not only to wash the chlorine, but to afford an indicator of the progress in the generator. The bottle must show a greenish gas, and the bubbling should be lively; should this

not be the case, the remedy is more sulphuric acid, until the last of the allowance is used up ; afterwards steam is turned on in the tub to heat the contents of the generator, and from time to time the contents are stirred through the large opening to prevent caking. The vat, after the pyrites has been sifted in as described, is left uncovered until the gas reaches the top of the charge, which should be within 3 to 6 hours ; progress is ascertained by the odour of samples, taken by hand, from beneath the surface. When the gas has been within an inch or so of the top, the cover is brought down and luted on with linseed-meal ; no escape of gas is allowable, as the respiration of chlorine is not only injurious but very disagreeable, so the cover must be quite impervious. A small hole closed by a cork is opened from time to time, to ascertain if the gas is up to the top ; this is done by presenting a glass rod dipped in ammonia at the hole, when, if chlorine is present, copious white fumes are given off. The chlorine is now permitted to operate upon the gold for 12 to 16 hours.

“After the allotted time, the cover is unluted and raised by block and tackle. A spout affixed to a piece of quartering is placed on the vat, and the hose is turned into it. When the vat is full, and no air-bubbles appear, the outer leaden tube is opened, and the water of lixiviation is allowed to run from under the false bottom into 30-gal. earthenware pans, care being taken that the hose delivers a quantity of water sufficient to keep the vat full, so that as much may be replaced as runs out below. The flow is allowed into the earthenware precipitating-pans until about 90 gal. has run after the last reaction was obtained in a test-tube with sulphate of iron ; the supply through the hose is then stopped, and the vat is allowed to drain until the time comes for throwing out the tailings.

“The precipitating-pans are of glazed earthenware, made at a local pottery ; they are twice the height of their diameter, and have a capacity of 30 gal. ; 5 stand on a turn table, on a truck, and their tops are about 6 in. below the bottoms of the vats. After the addition of sulphate of iron, they remain undisturbed all night ; in the morning the clear liquid is syphoned into a drain, and runs away, and the auriferous deposit is concentrated into pans standing at a lower level.

“The sulphate of iron is made from scrap iron and dilute sulphuric acid in a series of earthenware pans ; some is made every day, and the clear solution from that made the week before comes into use.

“The quantity of sulphate of iron used is always in excess of that supposed to be required, and the test of sufficiency is made by filtering a small quantity of the liquid from the precipitating-vat, adding some sulphate of iron solution, and noting whether the mixture darkens in colour ; if it does, more sulphate of iron must be added. When all the auriferous precipitate has been collected into one pan, a fine spray being

used finally within the upper series of pans, this is allowed to settle; the supernatant liquid is then syphoned off, and the precipitate is thrown on filter-papers within large funnels, and filtered; the precipitate is well washed, dried in a hot-air oven, and finally smelted in Hessian crucibles, a little nitre, carbonate of soda, borax, and salt being used as fluxes.

"The tailings from the vats are thrown out with a shovel into a trough, and sluiced out of the building. This trough is laid down with coir (coco-nut fibre) matting to arrest any stray particles of coarse gold which have resisted chlorination; the gold obtained from this source, though small in amount per ton, reaches an appreciable quantity per annum.

"Remarks.—Blende is troublesome, on account of the difficulty there is in driving off the sulphur. Antimony sulphide is still more so, because it agglomerates, and fuses so readily. The presence of 6 per cent. of galena does not prevent the gold being successfully obtained; a higher percentage has not been experienced.

"American and German writers give 95 per cent. of the assay as the average yield from successful chlorination; but percentages are delusive, as the following will illustrate:—

oz. dwt.		oz. dwt. gr.		oz. dwt. gr.	
Assay 10	0 per ton	..	95 per cent = 9 10 0	loss 5 per cent = 0 10 0	
Assay 10	0 per ton	..	95 per cent = 0 9 12	loss 5 per cent = 0 0 12	

"In dealing with percentages, the work has been done equally well in both instances, which is absurd, as the loss in the first is equal to the total amount as shown by assay in the second.

"The condition of the pyrites being the same, the loss is practically constant, or about 2 dwt. per ton, no matter whether assaying 10 dwt. or 20 oz. per ton. Any increase in the loss is due to faulty condition, as coarse crushing, or lumps and accretions caused by partial decomposition having set in.

"The progress of the work is—2 vats are charged and 2 emptied every day.

"The process, from the time the pyrites goes into the furnace to the time when the gold is ready for sale, occupies 10 days on the average. Over 8 tons is the average per diem through a month's or 12 months' run.

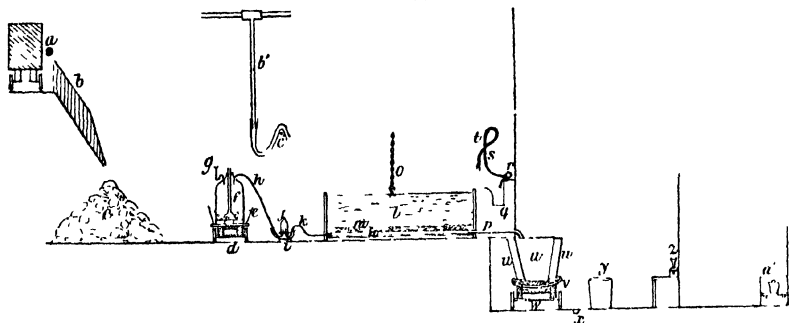
PRICES OF MATERIALS WITH CARRIAGE ADDED.

	£	s.	d.	
Manganese	6	0	0	per ton in the lump.
Salt	3	10	0	from neighbouring lakes.
Sulphuric acid	12	0	0	per ton (1'720 sp. gr.)
Labour, 35s. to 55s. per week.				
Carpenters, masons, and bricklayers, 60s. per week.				
Wages and supervision amount to about 20s. per ton.				

The apparatus is shown in Figs. 255 and 256. The reference letters indicate as follows. Fig. 255: *a*, truck conveying roasted pyrites out-

side the building; *b*, shoot to convey same inside; *c*, heap of roasted ore; *d*, truck for generator; *e*, tub in which generator stands; *f*, generator with stirrer; *g*, sulphuric-acid pipe; *h*, chlorine pipe; *i*, wash-basin; *j*, Winchester quart with bottom off; *k*, chlorine pipe; *l*, vat filled to

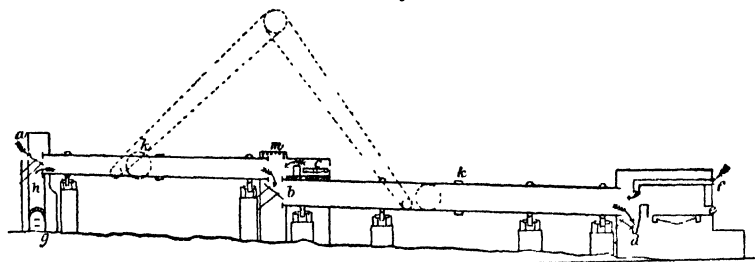
FIG. 255.



CHLORINATION AS CONDUCTED AT SANDHURST.

within 6 in. of top; *m*, filter bed; *n*, space under the false bottom; *o*, rope or chain for lifting the lid; *p*, pipe for conveying lixivium into precipitating-pans; *q*, sluice-trough; *r*, water main; *s*, indiarubber hose; *t*, peg to hang up hose; *u*, earthen precipitating-pans; *v*, turntable; *w*, truck for pans; *x*, drain; *y*, concentrating-pan; *z*, filter; *a'*, drying-oven and muffle; *b'*, steam-pipe; *c'*, hose. Fig. 256: *a*, where the pyrites

FIG. 256.



CHLORINATION AS CONDUCTED AT SANDHURST.

is fed in by a boy ; *b*, drop on cast plate from upper to lower furnace ; *c*, supplementary fire for starting the upper furnace ; *d*, where the roasted pyrites runs into a truck ; *e*, fire-place ; *f*, air-flue ; *g*, door for clearing the dust away ; *h*, flue leading to chimney ; *k*, cog-wheels to take driving-gear ; *m*, wrought-iron hood over the junction.

It has been pointed out by Küstel, Aaron, and others, that when lime or talc is present in the ore, salt must be added in the roasting, but they do not explain why. Experimenting on the cause, Burfeind found that

everything depends on the nature of the roasting. In a muffle, the charge does not get into contact with the fuel or products of combustion and the lime carbonate having been converted into sulphate remains so and is innocuous in the after treatment, the heat of such a furnace not sufficing to decompose the sulphate. But in a reverberatory or cylinder furnace, the charge does come into contact with the fuel or its gaseous products: the lime carbonate first becomes sulphate, then a part of the carbon is burned at the expense of oxygen in the sulphate, and lime sulphide is left. Here arises the difficulty. Unless this sulphide is entirely decomposed, the leaching is unsatisfactory. If chlorine or bromine comes into contact with calcium sulphide in presence of water, hydrogen sulphide will be evolved, and this gas will precipitate as sulphide any gold already in solution. Burfeind asserts that this is not re-dissolved by any excess of chlorine, but is lost in the tailings, notwithstanding others' statements to the contrary. Such gold can only be recovered by re-roasting and re-treating the tailings. Therefore Burfeind advises that in applying the Plattner process to ore containing lime, the absence of lime sulphide must be assured ere the material enters the leaching vat, and for such a purpose the ordinary ferricyanide test is not reliable.

Mears's process.—Mears's chlorination process is a modification of the well-known Plattner's chlorination process, differing mainly in the hastening of the combination of chlorine and metal by applying a concentrated solution of chlorine under pressure. Charges of pyrites of about 1 ton each are treated in revolving, air-tight, lead-lined, iron cylinders, 100 to 200 gal. of water being present with each charge. When the chlorination is complete, the fluid mud is poured into a filter, whence the gold chloride in solution runs to a precipitating-tank, where it is thrown down by iron sulphate.

In this process, the ore has first to be de-sulphurised by roasting in a reverberatory or other furnace. The chlorine required may be generated in whatever way is best adapted to the circumstances. When the chlorination of a charge is completed, the excess of chlorine gas in the chlorinating cylinder and that absorbed by the water is withdrawn for re-use. The chlorinated mass is leached to liberate the gold chloride in solution. The gold is precipitated from the solution either (*a*) by iron sulphate, with subsequent washing with sulphuric acid and smelting with borax, or (*b*) by filtering through charcoal, subsequently drying and incinerating the mass, washing out the ashes, and smelting as before. It is said that concentrated pyrites can be treated at a cost of less than \$5 (say 1*l.*) per ton in California.

When the process was first introduced, attempts were made to work by generating chlorine in the usual way and forcing it into the lead-lined

iron barrel by means of a pump ; but, as a pressure of about 40 lb. to the sq. inch was used, great difficulties were experienced. As to the advantages of the process, as now worked, Aaron thinks that, " unless in cases where it yields a notably higher percentage of the gold than by the old method, the game is not worth the candle. The great advantage of the old method is that it requires no moving machinery ; there is nothing to break down, and but little to wear out. A set of vats once installed will last a long time if properly cared for, and will consume nothing more than a little coal-tar and asphaltum, or paraffin, from time to time. Mere rapidity in working is of secondary importance ; the works of suitable capacity once started with a constant supply of material to treat, the same daily output is obtained if each charge takes 3 days as if it only took 3 hours to work. And I see no reason why vats of 50 tons' capacity should not be used in gold leaching as they now are in silver leaching ; also they might be arranged to be discharged by sluicing out the tailings, instead of shovelling, as is also done in silver leaching. Working 20 or more tons of material per day, with 50-ton vats, I doubt if the Mears process can offer any advantage, unless it be in an increase of the percentage extracted, in case of rather coarse gold, or that which contains a considerable proportion of alloyed silver ; in the latter case, the movement and friction in the barrel would do good by removing the film of silver chloride which would otherwise obstruct the action of the chlorine on the golden particles ; or, in case of material rich in silver, or lead, or both, like that which O. Hofmann found it necessary to treat by leaching with ' hypo ' before chlorinating the gold."

In the matter of precipitants for the gold, Aaron notes that ferrous sulphate generally answers very well, and where it does not, hydrogen sulphide supplies its place satisfactorily. The use of animal charcoal for this purpose, though very effective, and involving no trouble in the matter of settling, has not found favour ; there seems to be a difficulty in the separation of the gold from the carbon, the only feasible way being to redissolve the metal by means of aqua regia, and again precipitate it by some one of the known reagents for that purpose. The same applies in a less degree to the use of wood charcoal, though that is more easily disposed of by burning, the gold being recovered from the ashes by melting them with borax, &c. Sulphur dioxide (sulphurous acid) is an excellent precipitant for gold, which it throws down in the metallic state in the form of a dark powder ; it is easily produced by heating strong sulphuric acid and either charcoal or sulphur, in an iron retort, the resulting gas being passed into the solution of gold. There are many other substances which throw the gold down in the metallic state, and among them is one which Aaron thinks may be the best of all precipitants for practical use—that is precipitated copper sulphide. The copper sulphide is converted into

sulphate, which dissolves, while the gold is deposited in the metallic state, in a good form for collection. The copper sulphide may be recovered from the liquid by precipitation, by means of either hydrogen sulphide or an alkali sulphide, though in the latter case it would be mixed with some free sulphur, which would then become mingled with the gold, but could be easily burned off. The copper sulphide may be applied by stirring it into the gold solution until a test shows that no gold remains dissolved, but a better way would be to let the gold solution flow slowly through a series of small filters containing the sulphide. When the copper sulphide in the first filter is almost entirely replaced by gold, that filter must be emptied, refilled with copper sulphide, and replaced as the last of the series, the former second becoming the first, and so on. The precipitated gold may be freed from remaining copper sulphide by digestion with a warm, strong gold solution. This method offers advantages in the collection of the gold, which is thrown down in a granular condition, and when washed and heated, assumes the golden color. Any remaining traces of copper sulphide become oxidised in the heating, and may be removed by a little nitric acid, leaving pure gold for melting. Aaron thinks enough gold is lost, in most works, by imperfect settling, to pay the cost of this method of precipitating. In using iron sulphate for precipitation, Aaron found that after 48 hours' settling there remained gold in suspension to the amount of $\frac{1}{2}$ dollar for each ton of ore treated. The settling is promoted by the addition of some sulphuric acid to the liquid, also by repetition of the stirring about 2 hours after precipitation. When the ore contains copper, the liquid from the precipitation vats is conducted to other vats containing scrap iron, which precipitates the copper. The cement copper thus obtained always contains gold, and this appears to have produced an impression that iron sulphate does not precipitate all the gold from solution. Aaron's experiments have shown that when a sufficient quantity of iron sulphate is added, no gold remains dissolved; the gold found in the cement copper must, therefore, be that which had remained in suspension, and it indicates that, in case there is no copper to precipitate, the loss from this cause may be considerable.

Nelson Ferry recommends the addition of molasses to the leach when lime is present. He says: "Dissolve 1 gal. molasses in 30-40 gal. water and keep for use. The quantity to be used must be determined in each case by a laboratory test; if calcium sulphate comes down, either the molasses is in insufficient quantity, or it has not been thoroughly mixed. Examine by transmitted light. Avoid large excess of ferrous sulphate." If the gold comes down at first in a flocculent form, it is no matter; it soon assumes the usual form. The best results are got when the liquid is made slightly acid. The usual practice in such a case is to add sulphuric acid to the leach, and let it stand for a number of hours, then

transfer the liquid to another vat, and precipitate the gold with iron sulphate; the gypsum crystallises on the side of the first vat. This method requires a double set of precipitating vats.

A modification of the Mears process is applied to the treatment of some auriferous iron ores (see p. 665).

Ireland has published some general remarks on the application of Plattner's process, from which the following is condensed.

*The roast is the most important part of the process. The point to be attained is the complete oxidation of the baser metals, whose soluble salts would become precipitants for gold in the leaching vat. If the concentrated sulphurets have become lumpy, it will be necessary to recrush them, that the roast may secure complete oxidation. It frequently happens that too much material is put into the furnace at one time. Aaron advises, for the reverberatory, 10–16 lb. for every sq. foot of hearth. The furnace should be fired up several hours before being charged with the dry sulphurets, and the heat should be of a degree just sufficient to ignite the sulphur after the material has been evenly spread over the hearth. Too great a heat will matte the easily fusible sulphides, thereby necessitating a withdrawal of the charge from the furnace and a repulverisation. The presence of galena requires a low temperature and continual stirring. Deetken discovered that the presence of lime and magnesia, owing to their affinity for chlorine, were a detriment to the chlorination of the roasted ore, but he overcame this difficulty by an addition of salt to the charge in the furnace toward the completion of the roast. The roasting in all cases should be done at a gentle heat until the sulphur ceases to burn, which point is indicated by the disappearance of the characteristic blue flame, and the absence of scintillation on being raked. The heat is now increased and held at a high temperature until a thorough oxidation of the baser metals takes place. This can be ascertained by agitating, in a glass, a small quantity of the roast with water, allowing to settle, and adding to the clear decantation a few drops of the solution of potassium ferrocyanide. If a dark blue precipitate is given, iron sulphate is present; if brownish red, it indicates the presence of the corresponding copper salt; if both copper and iron are present as soluble salts, the two colours will unite, forming a dirty greenish-blue mass; in either case the oxidation is incomplete, and the roasting must be continued until a further test does not give any of the described reactions. Many operators continue the roasting to the oxidation of the iron only, but although it takes a longer time and a greater heat, it is advisable to also destroy the copper sulphate, as soluble metallic salts in the leaching vat react upon the iron oxide, which in turn precipitates the gold. If the concentrated sulphurets contain silver, the Paterson and Plattner processes should be combined. The silver is chloridised in the furnace by the

addition of salt during the roasting. After the gold is washed from the ore, the silver chloride is leached out by a solution of hyposulphite of soda. On the completion of the roast, the ore is drawn from the furnace to the cooling hearth, and, after a time, is sprinkled with water until moist enough to adhere when pressed in the hand.

In Europe the leaching is done in earthen or porcelain jars or tubs, wooden vats being condemned on account of their absorbing power. Deetken overcame this difficulty by coating the vats on the inside with pitch and tar. The damp roasted pulp is sifted into the vat and spread evenly and loosely until within about three inches of the top. The chlorine gas is now introduced through an opening in the bottom of the tank until it rises to the surface of the ore, which will be known by dense white fumes of chloride of ammonium being given off when a rod or stopper moistened with ammonia is held close to and above the charge. At this point the cover is put in place and carefully luted with some suitable tenacious substance that will not crack in drying. The chlorine is continued until the space above the ore is filled with the dense yellowish-green vapour; then the plug hole in the cover is stopped, and the vat and contents are left at rest for 1-2 days, at the expiration of which time the gold should be converted into a soluble chloride. Before leaching, the plug in the cover should be removed and the ammonia test applied. If there are no resulting white fumes, the ore must be again subjected to the chlorine, and so continued until the ammonia reaction is given. The chlorine is generated in a leaden vessel from a mixture of common salt, manganese peroxide, sulphuric acid, and water. The reaction is: $2(\text{Na Cl}) + 2(\text{H}_2\text{SO}_4) + \text{Mn O}_2 = \text{Na}_2\text{SO}_4 + \text{Mn SO}_4 + (\text{H}_2\text{O})_2 + 2\text{Cl}$. Care must be taken in using the chlorine gas that no muriatic acid enters the leaching vat, for, beside attacking the oxides of the metals, should there be an imperfect roast the sulphides would be attacked, eliminating sulphuretted hydrogen gas, which would precipitate the gold from the chloride solution. The chlorine is deprived of muriatic acid after leaving the generator on its course to the vat, by passing through water.

The ore being thoroughly chlorinated, the cover is removed and water is run into the vat until it rises above the surface of the mass. After a few minutes, when the water has had sufficient time to permeate all the ore, and still covers the surface, the plug at the bottom of the vat is removed, and the liquor is conducted to the settling tank, into which has been put the precipitant for gold—a solution of iron sulphate. A continuous flow of water must be admitted into the leaching vat: at no time during the operation, which lasts several hours, should the water be permitted to fall below the surface of the ore. At intervals a glassfull of the liquor, after having left the leaching vat, should be taken and

thereto a little of the iron sulphate solution added ; if gold were present a bluish-black precipitate would be given. In the event of any precipitate resulting from the unity of the two solutions, the washing is not finished, but must be continued until the precipitant added to a freshly-taken portion of the liquor does not give any reaction, or at the most only a faint coloration. After a thorough stirring of the contents in the settling tank, the precipitate is allowed to subside, and a glassfull of the clear liquid taken, to which is added a little of the iron sulphate solution ; if a dark coloration is given, more iron sulphate must be added to the liquor in the tank. Or, if on the addition of a few drops of gold solution to a little of the clear liquid from the tank no reaction is given, it is a confirmation of the former test that there is a deficiency of iron sulphate. Should the roasted ore contain silver chloride, it is leached with a solution of soda hyposulphite after extraction of the gold by water. The liquor is run into a tank set aside for this especial purpose, in which the silver is thrown down by a solution of calcium sulphide, as sulphide of silver. The outflow of the silver solution should be conducted slowly, by closing at intervals the discharge pipe, as the silver salt is not so readily extracted by its solvent as is the gold.

The presence of silver in the hyposulphite is known by the sweet taste of the liquid ; but when it can be no longer detected in this wise, a little of the outflow is caught in a glass and a few drops of the calcium sulphide solution are added ; if a dark precipitate is given, it is reasonable to suppose that the silver has not been entirely extracted, although other metallic salts, giving similar reactions, are also washed out by the same solvent. The only recourse at this point is to apply the confirming tests for such soluble compounds, as it is almost impossible to decide by the coloration which metals may be present. After the silver precipitate in the settling vat has subsided, a little of the calcium sulphide is added to the clear solution, and if a dark coloration is the result, it is evident that more of the precipitant must be added. After the tests have shown a thorough precipitation of the gold and silver, the tanks are left undisturbed until the suspended precipitates have settled.

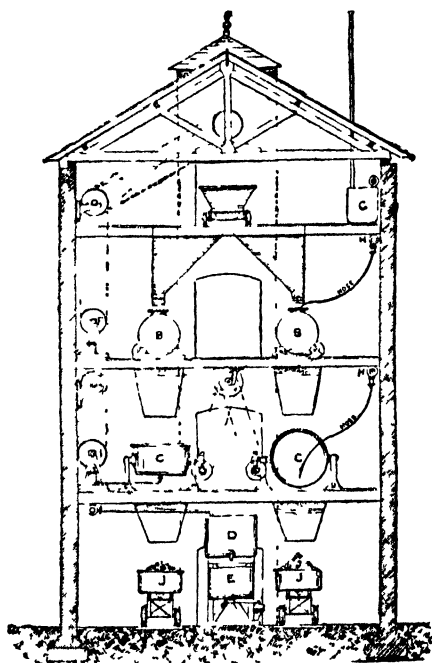
The silver precipitate will settle in a few hours, when the clear solution may be drawn off, the silver sulphide collected, conveyed to a filter, washed thoroughly with hot water, pressed, dried, and then roasted in a furnace—first, at a heat just sufficient to drive off the greater part of the sulphur ; at this point the temperature should be increased to a dull red, and so continued for several hours, or until the greater part of the silver becomes metallic. The partly-metallised substance is placed with borax and iron scraps in a graphite crucible and fired until thoroughly melted ; enough iron should be added that it does not all disappear in the operation. If an iron rod, immersed in the molten mass and allowed to

remain a few minutes, shows no signs on removal of having reduced in size—the iron scraps should be removed, the matte and slag skimmed off, and the metal poured.

The contents in the gold tank should remain at rest for at least 40 hours, for under that time a complete precipitation and settling of the gold cannot be ensured. When the gold has all settled at the bottom of the tank, the clear liquid is drawn off, the gold is collected, filtered, washed several times with hot dilute hydrochloric acid, and lastly with hot water, until all traces of acidity are removed. The gold is now dried, melted in a graphite crucible with borax, skimmed, base metals in a measure removed by the addition of nitre a little at a time, reskimmed, and poured. (Report, State Mineralogist, California).

Newbery-Vautin process.—The plant necessary for carrying out the process will, from local conditions, vary, but Fig. 257 shows an elevation

FIG. 257.



NEWBERY-VAUTIN PLANT.

of a 4-barrel plant and accessories, arranged within a building, the roasted ores coming into the chlorination mill in drop-bottom waggons, each of which has a capacity to fill one barrel. These waggons run on a tramway from the furnace to a power-lift at one end of the building; this raises the waggons and load on rails running over hoppers; the principal other parts are chlorinating barrels B, leaching-vat and filter C, reservoir for the liquid from the filter D, and charcoal collector E. The chlorination barrel is made of iron, lined with lead, this again with prepared wood; it is sufficiently strong to bear a pressure of 100 lb. to the sq. inch. In form it is cylindrical, and in the barrel are two doors directly opposite to each other. One door is sufficiently large to allow the

pouring in of a stream of tailings or ore; the other is smaller, and is for the introduction of compressed air. The barrels vary in size according to the quantity of ore required to be treated. The charge falls from the hopper through the larger door, when the required amount is poured in; chlorine producing chemicals are added (commonly lime chloride and sulphuric acid), and sufficient water is introduced so that the mass

may assume a semi-liquid state. The cover is then screwed down, and the barrel is turned half round, so that the small valve is uppermost. To this is attached a rubber pipe, leading from an air-pump F, and compressed air is forced in until a pressure of about 60 lb. to the sq. inch is reached ; then the valve is screwed down and the hose is disconnected. By means of friction-wheels (on which the flanges of the barrel rest), the barrel is set revolving at a speed of about 10 revolutions per minute ; this brings the lime chloride and sulphuric acid into combination, and produces the gas (chlorine), which, under the pressure now in the barrel, becomes liquid and mixes with the water, so that a strong solution of chlorine is in a continual state of agitation through the ore. The gold contained in the ore, from its contact with the chlorine solution, forms into a chloride of gold, which, being soluble in water, is rapidly absorbed by the liquid in the barrel. The time occupied in reducing the gold to this state varies according to the size of the gold grains, and the nature of the ore ; with very fine gold, 1 hour should be sufficient, but in cases of unusually coarse gold, 4 hours may be necessary. The chlorinator being stopped, the hose is again attached, and the compressed air, mixed with gas, is allowed to escape into lime-water tank G, so as to prevent the inhalation of any disagreeable fumes. The large cover is then removed, and the barrel is set revolving as before ; at every downward turn it discharges a portion of its contents into a shoot leading to the leaching-vat B below. When almost all its contents are discharged, water is thrown in from the main, which, by the revolution of the vessel, is swirled around, thus completely washing into the filter all remaining solution. The filter C is an iron vessel lined with lead, with a double bottom ; to the lower part of this a pipe is connected, leading to a vacuum pump I. When the ore from the chlorinator has filled the filter, the air is drawn from between the double bottoms of the latter, which causes very rapid filtration. The filtered liquid is drawn off through a pipe into a vat ; a continuous stream of water is kept playing upon the tailings in the filter, which, by the suction, is drawn through them, so that all the chlorides are washed out. To prevent any clogging of the material, the action of the exhaust-pump is made intermittent. The solution as it runs to the vat being continually tested, it is easy to know when the work is completed ; then the water is cut off, and the pump is stopped. The stuff in the filter is tipped into a truck J on a line of rails below, by which means it is run to the tailing heap.

The liquor drawn from the filter C contains gold in solution, lime, magnesia, copper, zinc, &c., according to the nature of the ore ; also, in all instances, it retains a greater or less proportion of free chlorine. This being a deterrent to the decomposition of gold chlorides, it becomes necessary to dispel it, which is done by blowing air or steam through the

liquid. If the water is entirely free from gas, decomposition of the gold chlorides occurs very rapidly when passed through charcoal. At the bottom of the vat D a trap K is fixed to run the liquid slowly through a filter of charcoal; contact with this material returns the gold chlorides to metal, they are so deposited amongst its pores and fibres, and on its surface. Copper, lime, magnesia, and zinc, are not touched by this reagent, so they pass through in a soluble state. The charcoal may be then burned in a crucible with borax, and the ingot of metal obtained. In some places it is difficult to arrange for a continual supply of charcoal; in such a case, the fine metallic gold may be dissolved from the charcoal by a weak solution of nitric acid and chloride. Such treatment very rapidly leaves the reagent clean—capable of being used again and again—and the gold is once more carried off in a soluble state by the liquor. The solution being now free of any substances but gold and aqua-regia, a steam or air blast is introduced into it, so as to gather up and dispel the nitric acid, and leave the gold chlorides perfectly free; then a solution of iron sulphate being added, the precipitation of the gold readily follows.

The usual size of the barrels is for a charge of 25 cwt., and the leaching pan, &c., are of equal capacity. This is found a convenient charge, and the cost of the whole 4-barrel plant, as shown in Fig. 257, with shafting, pumps, tanks, belts, lift, 6 waggons (3 for the top floor, and 3 for spent ore), including all patentees' royalties, and packing and delivering same f.o.b. is 1750/. Or, for a single barrel with driving gear, leaching-pan, or filter, tank for liquid from same, tank for absorbing gases from barrel, air-pump, vacuum-pump, shafting, pulleys, belts, pipes, one drop-bottom truck, one all-round tip waggon, small weigh-bridge for trucks, &c., complete, including royalties and delivery f.o.b. London, 420/. From the two foregoing the cost of any size plant may be easily arrived at. Sometimes it is desirable, where the plant must be placed in light timber or iron sheds that would be far too light to carry any machinery, to fit the whole plant up on a timber or iron framing, totally independent of the building which covers it; in fact it may, in some climates, be set to work in the open.

The preceding description is furnished by the licensed makers of the plant,—Calvert, Cornes, and Harris, 30, Walbrook, London.

Following is an account of the working of the process at Edwards's United Pyrites Works, Pinchgut Gully, Sandhurst. The pyrites is first roasted in the ordinary way, as for the amalgamating process. The machinery consists of a revolving barrel made of iron and lined with lead, the size of which is 4 ft. 6 in. diameter by 5 ft. in length. This is worked by a belt from an intermediate shaft from the engine. To this shaft 2 pumps are fixed, one an air, the other a vacuum. There is an

iron leaching pan lined with lead under the barrel, to receive the ore or sand when treated here. This pan has a false bottom fixed in a bed of quartz sand, which acts as a filter bed. There are also 2 liquor vats, each 5 ft. in height by 4 ft. 6 in. diameter. Besides, there are several earthenware pots or pans. The treatment of the ore is commenced by putting about 1 ft. water into the barrel with about 34 lb. sulphuric acid. The barrel, which contains about 25 cwt. of ore, is now charged, and about 30 lb. lime chloride are added. There is a cover to the barrel, similar to that of a retort used at gasworks, which is screwed down on to a rubber joint. A valve attached to the barrel is then opened to admit the forced air from the pump before mentioned. The amount of air admitted varies from 15 to 30 lb. to the sq. inch.

The barrel is now set in motion at a speed of about 7 revolutions a minute. The gear is frictional and can be thrown in and out at will. The barrel is kept revolving about 3 hours, more or less, according to the fineness or coarseness of the gold ; but, from assays made of the different charges treated, it is considered that a shorter time will suffice. The gold by this time is in solution. The revolving barrel is now stopped and the cover is taken off. When this is done, the barrel is again set in motion, and, being self-emptying, the ore falls on the bed of sand in the large pan before described. While the barrel is emptying itself, the chlorine gas smells very strongly and disagreeably, but it is said to be innocuous. The liquor or gold in solution is carried through this filter bed, while the sand remains on the top. The filtering is facilitated by the vacuum pump, which is connected to the chamber under the filter. The auriferous sand or ore now treated is removed by trucks. The vacuum pump, which may be also called the liquor pump, not only draws the liquor out of the leaching vats, but it also conveys it into the 2 vats mentioned. The sand or ore before being removed is washed with water until no reaction is given by the addition of iron sulphate to the liquor. In these vats are placed coils of perforated leaden pipes. They convey air into the liquor from the air pump, and also force out the gases. The liquor is next conveyed through a short rubber hose into charcoal filters. These filters may be described as earthenware pots of different diameters, containing columns of charcoal of about 4 ft. in height. The liquor or gold solution passes through the columns, and the gold is precipitated in its metallic state throughout the charcoal. The charcoal is then burned, when the ash is mixed with a little borax and then smelted. The water which passes out from the bottom of the columns is tested from time to time, to ascertain if it contains any gold.

The steam power required to drive the barrel is only nominal ; in fact, several more barrels could be worked with the steam power now in

use. The plant at present erected is capable of treating about 35 tons of ore per week, and parcels of pyrites have been received at the works, not only from other gold-fields in Victoria, but also from the other Australian colonies. It is estimated that an average of 93 per cent. of the contained gold is extracted by the process.

It thus appears that the process has fulfilled all that was expected of it in Australia. Readers of these pages will decide for themselves whether it possesses any advantage over the Plattner process so generally and long used in America, either in cost or result.

Pollok process.—This is another recent development of the chlorine principle. The ore is first roasted, as is usually done in all chlorinating processes. Even with float gold ores in which there is little or no sulphur, roasting is usually desirable, for it oxidises the iron and opens up the ore; but in a few cases, where the ore is already of a spongy character, and the iron completely oxidised, roasting may be dispensed with. After roasting, the ore is allowed to cool, and mixed with a small proportion of bleaching powder and soda bisulphate; then thrown into a revolving iron cylinder lined with lead and wood, similar to those in use. The cylinder is then closed, and the pipe from the hydraulic accumulator or pump is turned on, when water is forced in at a pressure of 100–120 lb. per sq. inch; the air being allowed to escape by a small stop cock on the cylinder. When the cylinder is revolved, and the contents thus mixed, the soda bisulphate acts on the bleaching powder, liberating chlorine gas, and the high pressure immediately forces the chlorine into the pores of the ore, and dissolves the gold as chloride. As soon as the gold is dissolved, which by this process takes $\frac{1}{2}$ –1 hour, the excess of chlorine is blown off, and re-converted into bleaching powder for subsequent use. The ore and liquid are then thrown on to a filter, and the liquid, which now contains practically all the gold, is rapidly filtered off and run into tanks by means similar to those in use. Ferrous sulphate is then added to the liquor, when the gold is precipitated in the metallic state, after which the liquid can be siphoned off, and the gold collected, dried, and fused into bars. When it is desired to chlorinate hot, which is occasionally beneficial, this is easily effected by allowing the ore to only partially cool before introducing into the cylinder.

Although hydraulic pressure is the essential feature of Pollok's plan, the use of soda bisulphate is also important, for it is a harmless salt, and, it is stated, does quite as well as sulphuric acid, which is well known to be a most corrosive and objectionable liquid to handle; indeed, it would be highly inconvenient to convey it to many gold fields, whereas soda bisulphate may be packed in barrels and taken without any difficulty.

Claudet's process.—This process is widely adopted for extracting the metals from the residue of Spanish cupreous pyrites, used in the manu-

facture of sulphuric acid. At Widnes, near Liverpool, it is employed in the following manner. (1) The residue (burnt pyrites) is roasted at a very low temperature, with the addition of common salt: the oxidation of the metallic sulphides and the decomposition of the sodium chloride give rise to the formation of sodium sulphate and soluble copper chloride, with the chlorides of gold and silver, soluble in copper and sodium chlorides. (2) The roasted mass is washed with water, to remove the soluble chlorides, including those of the gold, silver, and copper. (3) A weak solution of potassium or zinc iodide is added, to convert the soluble silver chloride into insoluble. (4) The precipitated iodide of silver is reduced by zinc plates, with formation of soluble zinc iodide, which is utilised for precipitating the silver in subsequent operations. (5) The reduced metals are melted, producing copper and auriferous silver.

By this process, the gold is obtained as a bye-product at a very small expense. But the conditions necessary for the success of the process are probably present in no gold-mining district: they embrace (1) an extensive manufacture of sulphuric acid, (2) an abundant and cheap supply of salt, (3) a large demand for sodium sulphate, and (4) a ready market for enormous quantities of iron oxide.

The liquors resulting from the leaching of the roasted pyrites contain the gold and the silver which has been dissolved in the first washing, the silver being kept in solution by excess of salt. These are collected on the Continent in a special vat lined with lead and painted on the outside with tar. This vat is either in series with the others on an incline, or above the others, and so arranged that it can communicate with any of the series of precipitation vats. Directly below it is a vat into which all the precipitates are washed and collected until there is enough to make it worth while to treat them. The silver liquors are first settled, and are then raised into this tank; their silver and gold contents being known, the proper quantity of zinc iodide, dissolved in water with some plumbic acetate, is introduced at the same time as the other liquors. Its object is to precipitate all the silver and gold. During the whole time that the tank is being filled, it is kept constantly stirred, to be certain of the intimate mixture of the iodide, and is then left to settle. This takes at least 48 hours, as the precipitate is light. In order to collect it quickly, at Oker, a mixture of glue and tannin, made by dissolving 600 grammes of glue in 10 liters of water, and adding to this 30–40 liters of tannin, made by boiling white-oak bark in water is added to the vat. In 24 hours the precipitate is settled, and the supernatant liquor is quite clear and ready to be drawn off. At Oker this is done through a lead pipe fastened to an upright board. The pipe is bent at right-angles at the bottom, the end being conical, and made of very hard lead. This fits into another hard lead cone attached to a pipe which passes through

the side of the tank. The two are screwed together by a nut, which has a number of threads, and is provided with a slot at the back to be certain of its pressing tight against the pipe. The upright board to which the pipe is fastened is attached to a lever filled with holes at short distances so that it can be attached to a pin in the side of the tank. The top of the pipe can in this way be secured at any level below the surface of the clear liquid, which is thus allowed to run off into the copper precipitating-vats. No attention need be paid to it until the top of the pipe gets near the precipitate in the bottom, when it must be carefully moved. When the tank is to be recharged, the pipe is made vertical when its top reaches above the top of the liquid. Generally the liquor is drawn off with a siphon, or through a hole in the side. In some works this precipitation is done only a few times, and the precipitate is then removed ; but in the Hartz, and generally, it is done as often as 25-30 times. It is then removed from the tank. At Duisbourg, where the precipitation-tank is above the others, this is done through a discharge-hole in the bottom at the side of the tank which, during the operation, is closed by wooden stoppers 2 meters long, which reach above the top of the tank. Through the stopper near the top, a hole is bored for a round piece of wood, which serves as a handle. The stoppers are removed, and the precipitate in the bottom is washed into the tank below. When this device is not adopted, the precipitate has to be first washed in the large tank and then scooped out, which takes a much longer time, makes the large precipitation-tank do the duty which could as well be done by a smaller one, and diminishes the output of the works by just so much. The sides of this tank are covered with gypsum, which is very rich in silver. When this crust is thick enough, it falls of itself ; the silver is then separated from it. The precipitate must be washed 7 or 8 times with water, to remove any trace of soluble material. It contains a considerable quantity of lead sulphate and chloride, a small quantity of the subsalts of copper, and the silver as iodide. The mass is kept stirred and pumped through a filter-press. The precipitate is reduced with zinc, to which some hydrochloric acid is added. The iodide goes to the zinc, which is used over again, and the other metals are reduced, but considerable quantity of the iodide is not recovered. The waste is made good by the addition of potassium iodide. The result of the process is a precipitate containing much lead and very rich in silver, holding almost all the gold. The amount of lead may be as high as 50-60 per cent. The silver will vary according to the richness of the ore, between 3 and 5 per cent. The gold will not generally be much above 0.05 to 0.06 per cent. This material goes to the silver works to be treated. The cost of the process is exceedingly small. In the Hartz, this precipitate was formerly treated

with sodium sulphide. The silver, with about 6 per cent. of copper, is transformed into sulphide. The whole is pumped through a filter-press. (Prof. T. Egleston.)

Henderson's process.—The Bede Metal Co. formerly* adopted the following method of separating the gold and silver from the solutions obtained in lixiviating the calcined copper-ores, according to the wet system originally introduced by W. Henderson. It depends on the fact that when sulphuretted hydrogen is passed through a copper solution containing a small proportion of silver, the latter metal is at first precipitated in much larger proportion than the former. In precipitating in this way 5 or 6 per cent. of copper from ordinary copper liquors, containing about 20 oz. of silver per ton of copper, no less than 80 per cent. of the silver is precipitated. Sulphide of copper is obtained, containing about 200 oz. of silver per ton of copper, and this is calcined and further treated for the separation of the silver, ultimately yielding a residue consisting of sulphates of lead and lime, oxide of iron, and chloride of silver, the last-named metal amounting to 8 or 9 per cent. of the total weight of residue. Henderson dilutes his solutions to 20° or 25° Tw., and adds a very weak solution of lead acetate, by which he obtains a cream-coloured precipitate containing about 53 per cent. of lead, 5 or 6 per cent. of silver, and 3 oz. of gold to each ton of precipitate.

Hollway's process.—Hollway's process for smelting the Spanish copper-pyrites is based on the utilisation of the sulphur in the sulphides for conducting their combustion, and the application of the sulphurous acid produced to the manufacture of sulphuric acid, or other purposes. The first condition is that the whole of the oxygen of the air driven into a thin stratum of protosulphide of iron (FeS) is utilised for oxidation; and the second, that by the heat evolved in the rapid oxidation of sulphides, and without the use of extraneous fuel other than that employed in producing a blast,—(a) about half the sulphur contained in the iron-pyrites (FeS_2) is expelled in the free state; (b) the remainder of the sulphur, excepting that left with the regulus, is principally evolved as sulphurous acid; (c) though only about 20 per cent. of sulphur is oxidised, the proportion of sulphurous acid to nitrogen is 14·9, or larger than that commonly obtained by copper-smelters who manufacture sulphuric acid; (d) volatile metallic sulphides, such as arsenic and lead sulphides, are distilled off with the sulphur; (e) copper, silver, gold, and nickel will be concentrated in the regulus, provided an excess of iron sulphide is always present. The quantity of coal necessary to produce

* Thomas Gibb, Esq., President of the Tyne Chemical Society, in an obliging communication to the Author, dated Dec. 4, 1880, states that this process is "now discontinued on account of local circumstances. Claudet's process is now the only one used for treatment of these liquors" in England.

the blast, calculated on the oxygen requisite for the oxidation which takes place, is $1\frac{1}{3}$ cwt. per ton of pyrites. The process seems chiefly advantageous for copper-smelters treating rich copper-ores, and who can utilise the sulphuric acid made.

Hunt and Douglas' process.—In the process invented by Prof. Thomas Sterry Hunt and James Douglas, the copper is removed from its ores in a dissolved state, the solvent employed being an aqueous solution of neutral iron protochloride and common salt. Most oxidised compounds of copper, when digested in such a solution, are converted into a mixture of protochloride and dichloride of copper, which are dissolved, while the iron of the solvent separates in the form of insoluble hydrous peroxide of iron. When the solution of the copper chlorides is brought into contact with metallic iron, the copper is separated in a metallic crystalline state, while the iron passes into solution, reproducing the iron protochloride, which is fit for use on a fresh portion of copper-ore. The reactions between the iron protochloride and the copper oxides may be thus expressed :—

For the protoxide : $3 \text{ Cu}_2\text{O}_2 + 4 \text{ FeCl} = 2 \text{ Fe}_2\text{O}_3 + 2 \text{ Cu}_2\text{Cl} + 2 \text{ CuCl}$.

For the binocide : $3 \text{ Cu}_2\text{O} + 2 \text{ FeCl} = \text{Fe}_2\text{O}_3 + 2 \text{ Cu}_2\text{Cl} + 2 \text{ Cu}$.

The metallic copper and protochloride react thus : $2 \text{ CuCl} + 2 \text{ Cu} = 2 \text{ Cu}_2\text{Cl}$.

The process is applicable to all ores of copper, but sulphuretted ores must first be roasted. Auriferous and argentiferous ores, chlorinated by this bath, may be subsequently treated either by dissolving the silver from the washed residues by a solution of sodium hyposulphite or chloride, or by amalgamation. The latter is preferable for ores carrying gold : such should be treated with the bath in a raw state, or after simple calcination. The process is very simple and inexpensive, is applicable to all ores of copper, requires no repeated addition of salts or acids, produces a superior copper, and needs only $\frac{3}{4}$ ton of iron for each 1 ton of copper produced.

Monnier's process.—Prof. Monnier's sulphatisation process is used at the New Providence mine, Nevada. The ore is hand-picked and run through Cornish rollers in the proportion of 80 per cent. of ore and 20 per cent. of sulphate of soda, until it passes through screens with 576 holes per sq. in. It is then fed into one of Bruckner's revolving roasting furnaces, 40 ft. long by 5 ft. in diameter, with a fall of 6 in. for its length towards the discharge end. The hearth is at the discharge end, and thus the flames meet the slowly descending ores, whilst the furnace revolves 3 times a minute. The chemical reaction in this furnace is as follows. The oxidation of sulphur produces sulphuric acid, which combines with the soda, forming a bisulphate of soda, and when this substance approaches the vicinity of the hearth, near its final discharge from the furnace, it is decomposed, and the bisulphate gives up sulphuric acid,

reacting on the sulphates and oxides that may have formed, at the same time converting into soluble sulphates silver, copper, lead, &c., iron alone remaining an oxide. The roasted ore is then placed in large tanks with water for "lixiviation" or leaching, and the liquor obtained is passed through layers of "cement copper" (pure) in other vats, in order to precipitate a portion of the sulphate of silver whilst the liquid retains a low temperature. The remaining solution is then run through an evaporating-pan, in order to regain the sulphate of soda for repetitionary use, and in this manner the process is continued until insoluble residues are obtained, which contain the gold and some silver; these are carefully ground in *arrastras*, and finally passed over electro-copper plates and through mercury wells, upon and in which the more precious metals are retained by amalgamation for subsequent retorting. The results obtained are said to be at the rate of 88 to 93 per cent. on the assay for gold, 70 per cent. for silver, and the whole of the copper. An advantage claimed for Monnier's process consists in the fact that in other processes the miners lose the silver during chlorination, and that chlorination of gold takes 48 hours on the average, whereas, by this manipulation, all the more valuable metals are collected, and this can be done in about half the time.

Paul's process.—The "electric dry amalgamating" process introduced by Almarin B. Paul is widely adopted in California. It is mainly as follows. The ore is first heated to dryness, and then reduced in the dry way by any suitable machinery to a fineness that will allow it to pass through No. 14 wire cloth. The crushed ore is next pulverised to fine flour in a barrel, and thence conveyed to an iron, wooden, or earthen cylinder, with the addition of 20 to 55 per cent. of mercury, the whole being shaken up together for about an hour. The process is said to be effective and cheap, and to entail only a small loss of mercury.

Washoe process.—The Comstock ores are divided into first, second and third classes, according to their treatment. The first class forms but a small proportion. The ore of this class is crushed dry, roasted with salt (a modification of Plattner's chlorination), and amalgamated. The ores of the second and third classes are subjected to the "Washoe process" proper, which consists simply in reduction by wet crushing, and subsequent pan-amalgamation, in the apparatus fully described in the preceding chapter.

Calcium Chloride process.—The ore is washed and ground to a fine powder, together with a little salt. This mixture is roasted in reverberatory furnaces till the sulphides, arsenides, and antimonides are decomposed. It is then teemed into wooden tubs, and treated with hot water, to wash out the copper and silver salts, and the oxides. The mass is next treated with a solution of 0.006–0.007 of calcium chloride in water,

mixed with an equal volume of hydrochloric acid of 1·002–1·003 sp. gr., until a sample of the solution gives no gold reaction when tested with an acidulated solution of stannic chloride. The solution containing the gold is heated in wooden bowls by steam to a temperature of 160° F., and precipitated by means of ferrous sulphate, sodium sulphide, or sulphurous acid, a small quantity of lead acetate being added during the precipitation to ensure all the gold being thrown down. This process has been employed at the Falee copper works in Sweden since 1885, for the treatment of tailings from 29,000 tons of copper ore, together with 1500 tons of gold ore. In 1886, the tailings from 14,000 tons of copper ore contained 41·82 gr. of gold per ton before treatment, and 4·04 gr. of gold per ton after treatment, the cost per ton being :—

	<i>d.</i>
Calcium chloride, 6·6 lb.	5
Sulphuric acid, 8·37 lb.	1
Lead acetate and reagents	$\frac{3}{4}$
Fuel for steam	1 $\frac{1}{2}$
Labour	1
Total	<u>9</u>

In the same year, 960 tons of gold ore contained 523·62 gr. of gold per ton, and after treatment 6·02 gr. per ton, the cost per ton being 12s. 2 $\frac{3}{4}$ d.

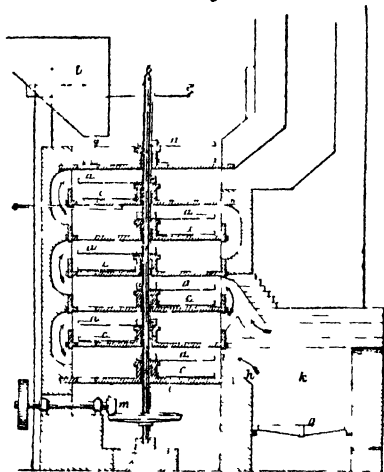
IRON ORES.—Iron has no effect upon the amalgamating qualities of mercury under ordinary conditions ; but mercury containing about 1 per cent. of sodium-amalgam will decompose iron salts, and produce what is termed “iron-amalgam.” Hence the use of too much sodium may defeat its own object by causing the mercury to take up (though only temporarily) the base metals.

In all gold-mining districts, immense quantities of iron-pyrites are produced. It has often been suggested that this pyrites should be treated in the same way as similar ores are dealt with in England for the production of alum, sulphur, and sulphate of iron. Stacks could be made of alternating thin layers of pyrites, aluminous shales, and fuel, and thus the gold would be left in the residue in a condition suitable for amalgamation. By simple exposure to the atmosphere and weather for a prolonged period, the pyrites undergoes considerable oxidation, with consequent liberation of the gold. Heaps of pyrites thus exposed are often sold to Chinamen, who make a good profit by their manipulation. But oxidation by natural means is necessarily a very slow operation, and several forms of furnace have been introduced for hastening it.

In Denny's self-acting drop-furnace (Fig. 258), the pyrites is delivered into a hopper on the top plate of the furnace. The plate has a self-acting grate, which, acting automatically, admits the necessary quantity

of pyrites on to the second floor. The arm *a* revolves with the shaft *b*, and the loose rake *c* sweeps the pyrites round the iron floor, thus presenting fresh particles of the matter under treatment to the passing oxygen. The top floor is for drying only. The next floor retains a charge of about 4 cwt. for 20 minutes, during which time it is continually stirred by the revolving arm, thus preventing caking, and ensuring free access of the oxygen to every particle of pyrites. At the end of the time named, the boy in attendance opens the hole in the floor by shifting a slide *d*, when the revolving arm sweeps the whole of the charge round, dropping it on the next floor. The operation of opening the slides and allowing the charge to drop is continued from floor to floor until it reaches the bottom thoroughly and completely oxidised. The furnace is built on a brick foundation, which supports the lower plate *e*. The next plate is 15 in. apart, and the others an equal distance. The shaft *b*

FIG. 258.



DENNY'S DROP-FURNACE.

passes through the centre of the plates, and is fitted with wrought-iron arms, which have loose rakes attached to them for sweeping the floors. The fuel is burnt on the bars *g*, and the products of combustion pass over the bridge *h* into the first floor, thence into the second, as indicated by arrows, and on the top. The free oxygen enters the hot-air furnace *j*, and passes into the third and fourth floor, combining with the base metals it comes into contact with. The carbon of the wood, having entered into union with its full amount of oxygen in the furnace, cannot take up any more. It therefore merely passes through the flues, heating the pyrites it comes into contact with, thus preparing it for entering into union with the free oxygen. The vertical shaft is driven by a pair of bevel-wheels *m*, and causes the whole of the arms to revolve. *k* is the fireplace, and *l* the hopper.

The ore of the Mount Morgan Mine, Queensland, consists of quartz and limonite with very finely divided gold enclosed. Experiments proved that the gold might be liberated if the ore was calcined at a heat just sufficient to dehydrate the iron ore, and on this an amalgamation process was devised. A practical trial proved, however, that there were other difficulties to be overcome, and that the loss still remained as high as 1 oz. or more per ton. The main cause of loss is one which may

account for unsatisfactory yields with other ores. It was found that in the calcination some of the iron in the ore was reduced in contact with the gold to the metallic state, and that an actual alloy of iron and gold was formed, which would not amalgamate with mercury. The Newbery-Vautin process (p. 652) was afterwards successfully applied to this ore.

The results of Stetefeldt's experiments on the Las Minas ores (p. 638) throw considerable light upon gold-amalgamation generally. It has often been found extremely difficult to extract a high percentage of gold from roasted ores by pan-amalgamation, and the theory of "rusty gold" has been extensively circulated to explain this fact. His experiments led him to the following conclusions:—(a) That the ore should be well roasted in as short a time as possible. The instantaneous roasting in a Stetefeldt furnace will, no doubt, leave the gold in a much more favourable condition for amalgamation than a long-continued roasting in a reverberatory furnace. Complete change of the magnetite to ferric oxide is not necessary, and even a large percentage of this mineral may remain unaltered without producing an injurious effect. (b) That copper, if it enters the amalgam, even in small quantities, prevents in a great measure the amalgamation of the gold. Lead will no doubt act still more injuriously. Hence, amalgamation in an arrastra, where no iron is exposed, will give far better results than if the process is conducted in an iron pan. Considerable copper sulphate may be present in the ore without materially impairing the result in an arrastra. (c) Amalgamation of finely divided gold requires friction, and contact alone with mercury is not sufficient. More than that: liquid mercury eludes in a great measure contact with gold, and it is necessary to amalgamate with gold-amalgam (or silver-amalgam in starting works on a large scale). This amalgam should not be too soft, so that, in the process of grinding, liquid mercury is not pressed out. In amalgamation generally, great fineness of the material treated is injurious. (d) Other conditions being equal, the result of amalgamation depends upon the grinding efficiency of the apparatus used. The arrastra is, under all circumstances, the best amalgamator, but must be so constructed that iron does not come in contact with the ore. (e) The small quantity of mercury reduced from mercuric chloride, by contact with iron under friction, produces at once solid gold amalgam, and the latter is really the effective reagent. That an arrastra, the bottom of which is covered with gold amalgam, is the most effective apparatus for amalgamating gold, has been known to the Mexicans for more than 200 years. If the Las Minas ores are roasted in a Stetefeldt furnace, and amalgamated in arrastras, there is not the slightest doubt in Stetefeldt's mind that 90-95 per cent. of the gold can be extracted. The process is well-suited to the

country, and the Mexicans understand its manipulations thoroughly. Metallurgical treatment on a large scale would comprise the following operations :—(1) Crushing the ore in a rock-breaker ; (2) drying the crushed ore ; (3) pulverising by rolls, and screening through No. 30 wire-cloth ; (4) roasting in a Stetefeldt furnace ; (5) amalgamating in arrastras ; (6) settling in American settlers, such as are used in silver mills ; (7) straining the mercury, retorting the amalgam, and melting the bullion into bars. At Las Minas is ample water-power, and ordinary labour and fuel are very cheap. Under existing favourable conditions, and with an output of 130 tons a day, the expense of mining and reduction per ton, would be :—

	\$	c.	£	s.	d.
Mining	1	27	0	5	3
Transportation	0	42	0	1	9
General expenses and skilled labour	1	19	0	4	11
Ordinary labour	0	41	0	1	8
Wood	0	21	0	0	11
Wear and tear of machinery, lubricants, mercury, &c. ..	0	90	0	3	9
Interest and amortisation on a capital of \$250,000 ..	0	78	0	3	3
Total expense, in gold	<u>\$5</u>	<u>18</u>	<u>1</u>	<u>1</u>	<u>6</u>

Hence ores of a very low grade can be worked with a profit.

The subjoined description of the Mears process (see p. 646) as conducted in Canada, is due to John E. Rothwell, M.E., Superintendent of the extensive works at Deloro.

The ore, an arsenical sulphuret of iron, after mining and hoisting, is hand sorted, sent to the mill, where it is crushed in rock breakers and rolls to pass through a No. 16 mesh ; then seized and run on the jigs for concentration. The concentrates are roasted in two large Howell-White revolving cylinders, arranged so that a preliminary roasting is carried on in the one, in which the arsenic and sulphur are nearly entirely driven off ; then the ore runs by gravity into the second cylinder, and is there roasted "dead" or "sweet." It falls from the lower end of the second cylinder to the cooling floor, and is there spread out. When cool, it is loaded into iron hopper cars and sent to the chlorinators. The furnaces have a number of important improvements upon the original furnaces, by which they are able to treat a very large amount of ore, and also without the usual heavy loss, and trouble with dust, which is always one of the chief objections.

The chlorinators are cylinders of $\frac{1}{2}$ -in. boiler iron, with cast-iron heads riveted or bolted on each end ; each head has a long trunnion, turned true, and acting as journal bearing and driving shaft. The trunnions are cored out, and have a shafting box and gland fitted to the end. The cylinders are lined with 20-lb. sheet lead, the joints all being "burned"

together. A chlorinator to hold a charge of 1 ton is 40 in. diameter by 44 in. long inside. One to hold 3 tons is 60 in. diameter by 72 in. long. A manhole is provided in the centre of the shell for repairs. In the manhole cover is the opening for charging and discharging the pulp, fitted with a light and strong lead-covered cover, arranged to be easily and quickly removed. On the inside of the cylinder, and on that side from the charging hole in the direction in which the cylinder revolves, is a lead-covered pocket, the lead of which is burned to the lining of the cylinder. The capacity of the pocket of a 1-ton chlorinator is about 100 lb., and a 3-ton, 300 lb. of sulphuric acid. Several lead-covered shelves are bolted to the inside of the cylinder for the purpose of stirring the pulp thoroughly.

Through the trunnion is passed the goose-neck, an iron pipe covered and lined with lead, and turned smooth and true where it passes the stuffing-box and gland. Inside the cylinder it is bent up, and the top end, which just clears the shelves, is turned in the direction in which the cylinder revolves. The goose-neck is held stationary in the upright position by clamps. The object of the goose-neck is to be able to draw off or pump the gas into the cylinder while it is in motion, and when a pressure gauge is attached the variation of pressure can be noted. The charging is done in the following manner: the car from the cooling floor is hoisted to a platform over the chlorinators; the charge of lime chloride is placed on the top of the ore. The water, about 120 gal. to the ton of ore, is measured into the cylinder first; then the ore from the car is let in, and after examining the acid pocket to see that no ore or lime has got into it, the sulphuric acid is poured in through a funnel; the cover is put on and fastened tight, and the cylinder is started. The proportions of bleach to acid used at Deloro were: Lime chloride, 6; acid, 7. The quantity of each varies with the quality and roast of the ore.

At the end of the time allowed for the cylinder to revolve—usually $1\frac{1}{2}$ –3 hours—the excess of gas is drawn off through the goose-neck, and an exhaust pump or ejector is attached to produce a partial vacuum, the cylinder revolving all the time. In this way most of the free gas is got rid of. Then the cover of the discharge hole is quickly removed, and the charge is dumped on a sand filter with a tight-fitting cover and exhaust pipe. Another method, much simpler and quicker, is washing the charge in the chlorinator. After the gas has been exhausted, the cylinder is filled with water and allowed to revolve a few times; the liquor is allowed to settle a short time, and then the supernatant liquor is decanted into a series of settling tanks; this is repeated until the liquor shows no gold on testing. In this way a charge of 3 tons can be thoroughly washed in 2–3 hours, while a 1-ton charge on a sand filter requires 6–8 hours.

The clear solution from the settling tanks and filters is run into the precipitating tanks. The precipitant is hydrogen sulphide gas, made from paraffin and sulphur in the following way :—The generator is a small cylinder of heavy boiler iron, with cast-iron heads riveted in, and all the seams well caulked. In one of the cast-iron heads is a hand hole, used to clean out the residues from the charge, and in the shell of the cylinder is the charging hole and a piece of 2-inch gaspipe, the outlet of the gas. The charging hole is fitted with a tight cover, easily removed. The generator is built in a small brick fireplace, leaving the charging hole and gas outlet on top free, and the discharging hole accessible from the outside. After charging with the proper proportions of paraffin and sulphur, a light fire is all that is necessary to generate gas. The best proportions, using single-pressed paraffin and common brimstone, are 1 : 2. The precipitating tanks are fitted with a tight cover, with an exhaust pipe leading out of the building for the escape of gas during precipitation, and a manhole with a tight cover. Near the bottom of the tank is a perforated lead pipe, one end coming up through the cover and fitted with a hose connection. In front of the precipitating tanks is a line of steam and gas pipes, fitted with T's, valves, and hose connections for each tank. The gas from the generator passes into a receiving tank, where the volatile oils, &c., that will condense, will collect. From the receiving tank, the gas is drawn by a small air pump and forced through the line pipes to the tanks and through the solution to be precipitated. To avoid creating a vacuum in the generator and gas receiver, a small air hole is left in the receiver. In this way the pump is able to work much faster, and, by driving air through the solution, materially aids the precipitation, and reduces the amount of precipitant used. The tank, after the precipitation is complete, is allowed to stand for a short time and then several small pressure-filters are attached, and the solution is allowed to flow through them ; 3 of these filters will empty a 1500-gal. tank in 3–4 hours and leave it ready for another precipitation ; or the precipitate can be collected in another filter press, pressed into cakes, dried, roasted, and melted with borax.

The ore treated is an arsenical sulphuret of iron, in a gangue of quartz and calc spar, an analysis of which gives about 42 per cent. arsenic, 20 sulphur, and 38 iron. An average assay value of a number of tons of the roasted concentrates treated in the chlorinators gave \$78 67, and the tailings assayed \$3 per ton, or 96·16 per cent. of the gold in the ore was extracted.

Thies contributes the following description of barrel chlorination, as in practice at the Phoenix Mines, North Carolina, and the Haile Mine, South Carolina. Its success he attributes to the generation of chlorine in the ore

pulp, to attrition, and to quick filtering. The original Mears process in working well-roasted ores was satisfactory ; but the leakage, the continued wearing out of goose-necks, the collapsing of the inner lead lining by a too strong exhausting of the gas, by being too often deceived as to the pressure of chlorine, when pressure, but no chlorine, was present, causing rehandling of the ore, made the process expensive and unsatisfactory. In this way he worked for nearly 2 years, testing, meanwhile, the effect of a highly saturated chlorine water under attrition without pressure. After repeated tests, he found his results equally as good without pressure as with the highest pressure, and better when he divided the requisite amount of lime chloride and sulphuric acid, so as to have nascent gas during the time of working. Thies then remodelled the chlorinator by closing up the hollow trunnion, removing the goose-neck and acid chamber, and substituting a lead valve connected with the inner lead lining in such a way that the pressure of free chlorine can be ascertained at any moment, and no charge subjected to chlorination need be thrown on the filter without full knowledge of the work being complete. The cast or sheet iron cylinder (chlorinator) is 42 in. diameter by 60 in. long. The heads are cast and securely bolted to end flanges, and provided with tight and loose pulleys. The bung for the introduction of the roasted ore and chemicals, 6 in. diameter, is provided with a lead-lined cover, which before rotation, must be closed hermetically. The interior of the cylinder is lined with sheet-lead of 10-12 lb. per sq. foot. The capacity of chlorinator is 1-1 $\frac{1}{4}$ tons of roasted ore. Before introducing the ore, the chlorinator is charged with 100-125 gal. water, or enough to make an easy flowing pulp. This done, the roasted ore is introduced ; half the requisite quantity of sulphuric acid is then poured in, and lastly half the required lime chloride, when the bung-hole is closed, and the chlorinator set in motion at the rate of about 15 revolutions per minute.

For Phoenix ores Thies uses 40 lb. lime chloride and 50 lb. commercial sulphuric acid per ton of roasted ore, but charges 20 lb. lime chloride and 25 lb. acid first, rotates for 3-4 hours, opens the bung, and charges the other half, having found better results by dividing the chemicals. On rotating for 2-3 hours longer, if, by the aid of the lead valve, free chlorine is found present, the cover is removed from the bung-hole and the chlorinated ton of ore is thrown on a shallow filter 6 by 8 ft. provided with a 5-inch filter-bed over which the pulp spreads to a thickness of about 4 in. The filter, before the ore pulp is thrown on it, is first flooded with clear water from below, and when the water stands over the filter the discharge hole is corked, so that the water acts as a cushion against the ore pulp, prevents the packing of the filter-bed, and admits of a free filtering.

When the chlorinator has been emptied on the filter, the cork is

removed and the solution is allowed to pass into a stock tank below. As soon as the first solution has passed through, so that the ore surface is exposed, 3-4 in. of water are added over the whole surface, and when this is filtered through and the ore surface is exposed again, the whole space above the ore, about 11 in. in depth, is filled, which, by practice on Phoenix ore, has proved sufficient to remove all the gold chloride; but should there be still a reaction with ferrous sulphate, more water must be added. The filters are lead lined, 18 in. deep, and have fall of 1 in. towards the outflow. The bottom is first covered with perforated glazed tiles of clay or of mineraline, which is impervious to the action of acids and chlorine. On this rests the gravel filter-bed, which is topped off with ordinary clean river sand. To prevent the filter from getting an uneven surface, longitudinal $1\frac{1}{4}$ -inch wooden slats 8-10 in. apart keep it in place. The filtering should be accomplished as quickly as possible; but as this depends generally on the fineness of the ore treated, no rule can be established. As long as the solutions show the presence of chlorine when the last wash water has passed through the filter, there is no danger of not having clean tailings.

The solutions accumulating in the stock-tank are let off into smaller tanks for precipitation with ferrous sulphate, which should always be regenerated if not active, so as to destroy any ferric sulphate which will dissolve precipitated gold [not in presence of ferrous salts]. Care should be taken, by examining after 24 hours, to ascertain if all the gold has been precipitated, as losses have occurred by a partial precipitation.

The precipitation tanks should not be too deep; a convenient size is 6-8 ft. diameter and 3 ft. high, holding the solution from about 3 tons of roasted ore. A sufficient number should be on hand to allow the precipitate at least 3 days to settle. After 3 days' settling in shallow vats, the supernatant liquor can be drawn off, and fresh solutions added for precipitation. At the Phoenix, the liquor is passed over metallic iron, and the copper is recovered as cement.

From the precipitating tanks the precipitate is finally collected, washed as clean as possible to remove the iron salts, dried, and melted. The amount of lime chloride and acid used at the Phoenix is due to the presence of quite an appreciable amount of chalcopyrite. An excess of acid should always be used, so as to convert all the lime into sulphate to remain in the filter. The solution should react slightly acid; if neutral, soluble lime chloride will cause a bulky precipitate with ferrous sulphate.

At the Haile Mine, where the ore is a pure iron sulphuret, Thies uses at present but 10 lb. lime chloride to 15 lb. of acid, and treats 4 tons of roasted ore in 2 chlorinators during 10 hours, and he has not failed to extract 94 per cent of the assay value. The Haile Mine ores offer no

obstacle to barrel chlorination, and are cheaper and easier to chlorinate than the Phoenix ores.

As the success of chlorination, by whatever process, depends on a thorough roast, assuming clean concentrates, it is of the utmost importance that the roaster should have some guide to go by, and to this end Thies lets him test every charge before drawing, by a bright filed iron rod. A small portion of the roasted ore is boiled in water and stirred with the bright iron; the least trace of sulphates will stain the iron—a sign for the workman that the roasting is not completed. At the Phoenix, Thies uses a revolving pan furnace of 12 ft. diameter, with a short reverberatory attached. From two working doors the roaster can rabble the ore. When a charge is finished, the ore is discharged, through the hollow axis on which the pan revolves, into an outer circle below, and is then removed by scrapers attached to the bottom of the pan, into a car, and delivered to the cooling floor, whence it is elevated into the chlorination house. Such a pan furnace roasts 1 ton of raw ore in 12 hours, with a consumption of $\frac{3}{8}$ cord of wood, and 90 cents for labour. The power necessary to drive the pan is a small item, and will not exceed 25 cents per ton of raw ore.

At the Haile Mine a double reverberatory furnace furnishes 2 tons of roasted ore every 24 hours, with an average consumption of 1 cord of wood at \$1.25 per cord, and 4 labourers. The cost per ton of roasted ore amounts to \$2.62½.

The cost of chlorination by the barrel process depends chiefly on the number of tons chlorinated per day: 2 men can easily chlorinate 4 tons in 10 hours, elevate the ore, and clean out the filters, of which Thies has 4 to each chlorinator, and having arranged on this basis the work at the Haile Mine, the cost for chlorinating 4 tons daily is as follows:

	\$	¢	£	s.	d.
40 lb. lime chloride, at 3 cents	1	20	0	5	0
60 lb. sulphuric acid, at 2 cents	1	20	0	5	0
2 labourers, at 90 cents	1	80	0	7	6
1 chlorinator	2	00	0	8	4
Motive power	0	50	0	2	1
Total	6	70	1	7	11
(Or \$1 67½c. (7s.) per ton.					

Add to this 12½ cents (6d.) for sulphuric acid for making iron sulphate, and 20 cents (10d.) for repairs and wear, which is more than liberal, and we have the sum of \$2 (8s. 4d.) per ton for chlorination, or \$4.62½ (19s. 3d.) for roasting and chlorinating 1 ton of roasted ore representing 1½ tons of raw iron pyrites. Inside of 7 hours from the time the ore is in the chlorinator, the solutions are ready for precipitation and the tailings are clean. The wear on the inner lead lining of the chlorina-

tors is imperceptible—a chlorinator in use at the Phoenix for over 5 years does not (Oct. 1888) show any wear on the lead. (Report, State Mineralogist of California, 1888.)

Gybbon Spilsbury adds that the Haile ores contain a considerable percentage of magnesia, which at first gave very voluminous precipitates. But the precipitation of lime and magnesia may be entirely prevented if care is taken to keep the tank-solution slightly acidulated. The moment the solution is allowed to become neutral, the magnesia begins to show in the precipitating tank as a milky cloud, which, however, immediately disappears on the addition of a small proportion of free acid. The cost of roasting, labour, chemicals, and power at Haile will never exceed \$4 (16s. 8d.) per ton. So far it has not attained that figure, some work having been done as low as \$3.15 (13s. 2d.) per ton. The actual gold recovered is in all cases within a few cents per ton of the full assay-value of the ore, less the amount shown by assay to have remained in the tailings. While the gold obtained in stamp-mill and amalgamating works is seldom purer than 897 fine, the gold obtained by this process is generally over 978 fine. (Trans. Amer. Inst. Min. Eng.)

In another communication to the Institute, Spilsbury deals with the general treatment of the similar ores of the slate deposits of Georgia and the Carolinas, showing the results attained by different processes adopted successively.

The Haile ore consists of a very siliceous talcose slate, from the body of which most of the magnesia has been eliminated and replaced by silica, but in the cleavage planes of which exist thin laminae of almost pure talc. The whole mass is mineralised with sulphides, mostly so fine as to be invisible to the naked eye. The gold also present is in an impalpably fine condition, both as free gold, and also as combined with the sulphides. These sulphides are entirely ferric, the faintest trace of copper appearing only once in a while. In addition to the finely comminuted sulphides in the mass of the slate, the veins are traversed by bodies of almost pure iron sulphides, some of which run very high in gold, while others again are entirely barren, and carry no gold at all. The gold is invariably in such a finely divided state, that it is impossible to detect the smallest speck by the naked eye, or even under the microscope. It is from this fact that the difficulty of the economical treatment of the ore rises.

For the first 2 years at Haile they were working practically above water-level, and at that time the ore averaged \$5-10 (1-2%) a ton, was practically free, and offered no difficulty to ordinary treatment. When they began to mine in depth, it was soon found that while assays showed about the same general average percentage of gold, the yield in the stamp-mill was greatly lessened. On investigation, Spilsbury found that

the percentage of sulphides was increasing, and that the gold appeared in a finer condition than in the surface-ores. From 30-mesh screens he changed to 40-mesh, and at once found great improvement. He has since then continued to use these screens; but in order to crush still finer, the discharge has been raised, so that now the pulp is of such fineness that 92 per cent. of it will pass through a 100-mesh screen. This has been done to be sure of freeing practically all the gold, which is proved to exist in the ore principally in the sizes from 70- to 90-mesh. The largest portion of the free gold is extracted by amalgamated copper plates, both inside and outside the mortars. The inside plates, which are removed daily and scraped, catch somewhat over 33 per cent. of the whole product of the mill. From the copper tables, the pulp passes on to a series of Embrey tables, also fitted with amalgamated (silver) plates. These tables furnish the concentrates for further treatment. Taking the average of the ore as \$6 (25s.), they generally extract \$3.20 (13s. 4d.) a ton as free gold on the plates. The pulp going to the Embreys averages therefore \$2.80 (11s. 8d.) a ton. The final tailings from the concentrators show 70-80 cents (2s. 11d.-3s. 4d.) a ton. The heads from the tables generally average \$35-37 (7l.-7l. 10s.) a ton. The necessity for fine crushing is shown by the fact that, before adopting the present fine screens and high discharge, while the pulp going to the concentrators would average nearly \$4 (16s. 8d.) a ton, the concentrates would never average over \$26 (5l.) a ton, while the final tailings ran \$1.10-1.30 (4s. 7d.-5s. 5d.) per ton.

The concentrates are then roasted "dead" in an ordinary double hearth reverberatory furnace. It is in the further treatment of these roasted concentrates that the principal difficulties have lain. For some years after starting, and while still using mostly surface-ores, Spilsbury introduced the Designolle process. At that time the concentrates hardly ever averaged over \$16 (3l.) a ton.

The Designolle process, whilst in vogue, was conducted as follows. To effect thorough roasting of the concentrated sulphides, a special furnace was designed, consisting of a large coil, 450 ft. long, of 5-inch wrought-iron pipe, built in a suitable furnace, and heated to a low red heat. The finely-divided ore is blown through this pipe, with a large excess of air, into suitable receiving-chambers. The roast made, when working right, is most complete, scarcely the faintest trace of sulphur being left in the ore; and further, through the constant attrition of the particles of ore among themselves in their passage through the pipes, the gold is entirely freed from all scale, and left perfectly bright and ready for amalgamation. Like all mechanical roasters, however, this furnace could not be relied on for perfectly uniform results; and Spilsbury fell back on the old-fashioned, but always reliable, *fort-*

schaufelungs furnace. Where labour is obtainable at 90 cents. (3s. 9d.) a day, and where wood costs less than \$1 (4s. 2d.) per cord delivered at the furnace, the advantages to be gained by automatic rabbling, in whatever way attained, are so slight that they are much more than counter-balanced by the perfect control one has all the time over the charge roasted in a reverberatory furnace. The advantages of the latter, in the matter of first cost, are undeniable. Double-hearth furnaces, 40 ft. long by 5 ft. wide, did not cost over \$800 (160*l.*) all complete, and had a capacity of 6-10 tons a day of 24 hours. Taking an average of 8 tons a day, the cost of roasting does not exceed \$1.10 (4s. 7d.) per ton.

The roasted ore, after cooling, then went to amalgamation by the old Freiberg barrel-process. The great objection to this is the length of time required for the thorough amalgamation of the pulp, owing to the difficulty of causing each particle of gold to come in contact with the metallic mercury, and the consequent excessive flouring of the latter. To obviate this, and cause a more thorough dissemination of the mercury through the charge, Designolle proposed using a solution of the bichloride instead of metallic mercury, at the same time placing in the barrel enough metallic iron to thoroughly effect the decomposition of the bichloride. The roasted ore, in charges of 600 lb. each, is put into a cast-iron cylinder or barrel, in which are also placed about 1000 lb. of cast-iron balls. The cylinder is then partially filled with water, and about $\frac{1}{2}$ gal. mercury bichloride solution is added. The barrel is rapidly revolved for 5 minutes, at the end of which time another $\frac{1}{2}$ gal. of the solution is added, and the cylinder is again revolved for 15 minutes. At the end of that time, every particle of gold has been amalgamated, and the barrel is discharged into a large settler, in the concave bottom of which is a bed of metallic mercury. The pulp is kept in suspension in this settler by revolving rakes, which maintain the surface of the mercury-bed clean and ready to absorb the particles of amalgam immediately they settle. The pulp is discharged from this settler in a thin stream over a series of revolving centrifugal copper plates, which catch whatever particles of amalgam may still be held in suspension. From the plate machine, the pulp discharges into a second settler similar to the first, this finally catching what may have passed through the former settler and the plates. The simplicity of the whole process is remarkable. With a plant costing in all less than \$4000 (800*l.*), including building, they treated 18-20 tons a day, with the labour of 1 man and 1 boy. The whole cost of treatment, including chemicals, labour, steam, and wear and tear of machinery, was 38 cents. (1s. 7d.) a ton.

The composition of the standard solution used was as follows : Mercury bichloride, 1 ; common salt, 2 ; muriatic acid, 1. The proportion of the acid may be varied according as the gold in the charge

to be treated is more or less rusty. These chemicals were mixed with water, in the ratio of 64 gal. to 8 lb. bichloride, thus giving .1 oz. bichloride to $\frac{1}{2}$ gal. of mixture. For all ores containing under $\frac{3}{4}$ oz. of gold per ton, 1 gal. of the mixture per charge is fully sufficient, after which $\frac{1}{2}$ gal. extra is added for each $\frac{1}{2}$ oz. of gold additional. The chemical reaction which takes place is $\text{HgCl}_2 + 2\text{Fe} = 2\text{FeCl} + \text{Hg}$. This simple reaction is due to the presence of sodium chloride in the mixture. Where this is not present, it is probable that the first reaction would be $\text{HgCl}_2 + \text{Fe} = \text{HgCl} + \text{FeCl}$, followed by a secondary reaction of $\text{HgCl} + \text{Fe} = \text{Hg} + \text{FeCl}$. It was therefore on the former reaction that Designolle based his invention. The extreme rapidity, however, with which the amalgamation takes place, seems to point to some further action than the above, as the decomposition of mercury bichloride by metallic iron is a comparatively slow process, and the globules of mercury might revolve for a considerable time in the mass, as in the old Freiberg barrels, without coming in actual contact with the particles of gold. In point of fact, however, it is just this slowness of action which is the chief element of success of the process. Very little, if any, of the amalgamation, Spilsbury believes, takes place by the particles of gold coming in contact with globules of free mercury; but when, in the acidulated bath, any particle of gold comes in contact with the iron, electrolytic action is at once established, and deposition of the mercury over the whole surface of the gold is instantaneous. It matters not how minute the portion of the gold exposed to the contact is, the whole surface is at once amalgamated. Even gold covered over with rust scales of iron, if exposed on one point only, will become thoroughly amalgamated throughout, the scale breaking loose, leaving the amalgam free.

The Plattner chlorination process was practically precluded by the impossibility of washing by filtration large quantities of impalpable iron oxide.

Spilsbury next turned attention to the concentration of the gold into an iron matte. But this proved unsatisfactory in experiments, except perhaps by addition of 15-20 per cent. of lime, which is totally absent from the rocks of the district, and too bulky to admit of being carried to the spot from any distance.

Nevertheless, from the researches of W. L. Austin, at Toston, Montana, it would seem that this method promises success in suitable localities. In some of the Hungarian smelting works, gold is concentrated by means of iron sulphide. On fusing the crushed pyrites (iron bisulphide) with lime, to flux the quartz, the pyrites loses half its sulphur, and becomes iron sulphide (FeS), which on fusion sinks below the slag, carrying with it the gold. Fusing raw sulphurets with lime has been tried in America with highly encouraging results.

The smelting of fine pyrites (concentrates, for instance) in a blast furnace, can be done by running the charge very low. The blast in such a case passes up through the charge freely, keeping the whole mass in a seething condition. Of course, dust chambers are a necessity when this is done. Or the reverberatory furnace might be used.

Prof. Clark thinks that, under certain conditions, gold and silver can be economically concentrated in an iron matte, making it possible to treat a certain class of ores not desired by the millman or lead smelter. His experiments on the Haile mine pyrites were not successful, and confirmed his opinion that, for successful working, some other metallic sulphide than iron must be present in the matte, and that the slag must contain some other base than iron. The fact that gold-bearing sulphurets are usually in a fine state does not appear to be a serious objection, as a partial roasting is in any case necessary to obtain the iron oxide to flux the siliceous ore. After roasting, a certain amount of iron sulphide is present, and if a little lime (5 per cent. or less) is added, and the mixture is air-dried, it will be sufficiently coherent to stand a moderate amount of handling, and will work well in a low blast furnace without giving an excessive amount of flue-dust. If both siliceous ore and pyrites be fine, it might do to mix them, partially roast and slag, and then smelt in the blast furnaces. If the *spur-ofen* type of furnace is used, and the slags are kept below 40–42 per cent. SiO_2 , the smelting should not be difficult. It has been suggested, since the experiments on the Haile concentrates, that if, after the charge in the reverberatory was thoroughly melted, a little raw iron or copper sulphide had been sprinkled on the charge, and the heat had been raised, it might have carried with it the gold in the slag.

LEAD ORES.—Various ores of lead, such as galena (sulphide), antimonial lead, sulphate, carbonate, arseniate, and phosphate of lead, are often met with in auriferous veins. The metallic lead derived from them has a highly detrimental effect upon the amalgamation process, causing a loss of gold-amalgam and mercury, through the lead-amalgam rising to the surface of the mercury as a frothy scum, carrying with it any gold-amalgam that may be present, and, by forming a coating over the mercury, preventing it taking up any gold that may pass over its surface. The lead-amalgam, when thus brought to the top, is easily broken up and carried away in a fine state of division by a stream of water passing over it. The whole of the lead-amalgam does not rise at once, and cannot be completely removed by simple skimming; but the more the mercury holding it is agitated, the quicker it rises. Metallic lead is sure to be reduced from any of the ores by the operation of roasting, and losses will therefore be increased in that way.

Austrian process.—At Lenz, in Austria, a fusion process is used for

treating the very poor ores of Zell, Rauris, and Boeckstein. These ores contain arsenical pyrites, the sulphides of antimony, copper, iron, lead, and zinc, and a very small proportion of gold and silver, alloying 15·33 gold and 84·67 silver. The operations carried out are as follows:—(1) Fusion for raw matte, (2) roasting of raw matte in stalls, (3) fusion without lead for a more concentrated matte, (4) roasting 2nd matte in stalls, (5) fusion with lead, (6) cupellation of rich lead.

The 1st fusion is in a furnace 24 ft. high, 3 ft. diam. of hearth, 4·5 ft. diam. of boshes, 2 ft. diam. of throat; 2 tuyères; $\frac{1}{8}$ to $\frac{1}{2}$ in. of mercury worked with a black throat.

The 1st matte averages in composition: iron, 55·1; copper, 4·3; zinc, 3·7; lead, 2·1; nickel, cobalt, arsenic, and antimony, 4·5; sulphur, 27·9; = 97·6. It contains 30 to 40 oz. troy of auriferous silver per ton.

The 1st matte is roasted 3 times in stalls containing 28 tons, and then worked with most extreme care in a furnace with a larger hearth and under much less pressure.

The 2nd matte is roasted as before, but with different conditions; when the hearth is full of melted matte, it is tapped, and the products run into a basin, where they are well stirred with poles. The matte is then partially taken off, the lead remaining until 600 to 700 lb. have collected. For a perfect extraction of the silver, it is necessary to charge 120 to 130 lb. of lead for each 1 lb. of silver and gold, which extracts 75 per cent. of these metals in one operation. The extraction of 75 per cent. of auriferous silver means that more than 90 per cent. of the gold and 73 per cent. of the silver has been obtained. A second operation removes so much more that, including amalgamation, where the loss is very great, more than 90 per cent. of the silver and 96 per cent. of the gold is obtained. This second operation takes place only when the matte is worked for copper. At other times, the gold and silver are obtained by charging the matte back in the first operation. If the 3rd matte is rich enough, it is fused with lead a second time; if it contains 35 per cent. or more of copper, it is worked for that; if under, it is roasted, and returned as a flux to the first fusion for raw matte.

Cupellation is performed in a German furnace of peculiar construction in a special way. Raymond recommends this process for the Colorado ores, which are of a similar character.

Richmond process.—The ore from the Richmond mine, Nevada, is put into a blast furnace, and the lead, which is present chiefly in the form of carbonate, runs out, carrying the silver and gold. This alloy is afterwards treated by a refining process, based on the Pattison process, but worked by steam, after the system of Luce fils et Rogan, of Marseilles, which separates it into soft lead free from precious metals, and a richer portion

containing all the precious metals ; the latter is cupelled in the usual way till all the lead is driven off, and an alloy of silver and gold is left in the cupel, to be run off into bars ("doré" bars, so-called), and sent to market as gold and silver alloys (see Silver).

SILVER.—All native gold is alloyed with silver, the latter often forming by far the greater proportion of the alloy. At Freiberg, the following simple plan is adopted for their separation. The alloy is collected in a granular form by allowing the molten metal to drop into water. The silver is then dissolved out by heating with double its weight of sulphuric acid in a cast-iron vessel in a reverberatory furnace. The gold remains undissolved, and sinks to the bottom. When first collected, it is still combined with much silver. It is washed with hot water, and again heated with sulphuric acid, until purified from silver and acid.

The gold and silver alloy obtained from the Richmond ores (see Lead) is granulated and thrown into a cast-iron vessel containing boiling sulphuric acid, which attacks the silver (when the gold contents do not exceed $\frac{1}{3}$ or $\frac{1}{4}$ of the silver), leaving the gold in a porous mass at the bottom of the vessel. The gold is collected, squeezed under a hydraulic press to remove the acid, melted, and refined. The silver solution (sulphate) is conveyed to a large leaden tank, in which are suspended copper plates ; by them the silver is thrown down, and is subsequently collected, squeezed, melted, and refined. The copper solution is then evaporated, and the blue copperas is collected in the usual way, to be sold to silver-mills, where it finds a use in the amalgamating-pans.

TELLURIUM ORES.—The ores produced by the mines of Nagyág and Offenbánya, in Hungary, and by some others, are remarkable as containing a comparatively large proportion of tellurium, which, although it has hitherto been of no commercial importance, has latterly been in some demand on account of a new application for the construction of thermo-electric batteries. The following experiments have been undertaken to discover a cheaper method of production than those heretofore in use.

The ore, as delivered for smelting, was found to be of the following average composition : quartz, 30 to 40 per cent. ; carbonate of lime, 10 to 20 per cent. ; carbonate and sulphide of manganese, 15 to 20 per cent. ; alumina, 5 to 8 per cent. ; galena, 5 to 8 per cent. ; copper-pyrites, 1 to $2\frac{1}{2}$ per cent. ; blende, 1 to 4 per cent. ; and small quantities of cobalt, nickel, antimony, arsenic, tellurium, gold, and silver.

When such a mixture of minerals is roasted, portions of the tellurium and gold are volatilised, and may be recovered in properly-constructed condensing-chambers. The manganese compounds are converted into manganic oxide, while the greater part of the gold is reduced, so that about 50 per cent. of the total amount may be saved by amalgamation.

By subsequent treatment of the roasted ore with weak hydrochloric acid, which can be done in wooden vats lined with lead, chlorine is generated in considerable quantity, through the action of the manganic oxide, and the whole of the valuable metals present, with the exception of silver, which remains in the insoluble portion, are converted into soluble chlorides. Any excess of chlorine produced in this operation is economised by condensation in water, which gives a liquor that can be used for redissolving the crude tellurium. The solution of chlorides obtained by this treatment is next cleared from lime and lead, which are precipitated as sulphates, by the addition of sulphuric acid. The separation of these sulphates is effected by subsidence and decantation, as filtration is found to present considerable difficulties.

Gold is next precipitated from the clear solution by the addition of a solution of sulphate of iron; and after filtration, tellurium, by the action of metallic zinc, which produces a black, muddy precipitate. This may, after washing with hydrochloric acid and rapid drying, be converted into crude tellurium by fusion, with any flux, in a porcelain crucible; but the product so obtained invariably contains lead, copper, nickel, and antimony, and it is therefore preferable to redissolve the first telluriferous precipitate in chlorine-water, and subject the solution for a considerable time to the action of sulphuric acid, whereby tellurium in a high state of purity can be obtained.

The original residue of the chlorination treatment contains, in addition to silver as chloride, some gold in a soluble state. By the addition of sulphate of iron to these residues when in a moist condition, the gold may be reduced, and the substance is then fit for treatment by amalgamation; but fusion with lead, when it can be done, is generally preferable.

The following results were obtained in an experiment conducted according to the above principle: 14.5 lb. of tellurium ore, containing 14 dwt. of gold and 13.9 dwt. of silver, were roasted for 1½ hour in a muffle-furnace. The loss of weight was equal to 7.2 per cent., and 0.35 per cent. of gold and 3.8 per cent. of silver were computed as lost by volatilisation. The roasted ore weighed 14.3 lb., of which quantity 13.2 lb. were taken for subsequent treatment by chlorine. This was effected by mixing it with 10.4 pints of water, 6.8 pints of crude hydrochloric acid (25° B.), and 10.6 oz. of concentrated sulphuric acid. The addition of the acid was attended with effervescence, owing to the rapid evolution of carbonic acid and chlorine.

After 24 hours, the solution was diluted by the addition of 6.8 pints of water; the whole contents of the dissolving vat were stirred well together and allowed to settle for two hours, when the clear liquor was drawn off. This operation was repeated three times, giving a total

quantity of 2 gal. of liquor, which was then treated with a solution of green vitriol (3·5 pints of 25° B.) in order to separate the gold. This was completely effected in 24 hours, and the resulting gold, after being washed, dried, and cupelled with lead, weighed 10·5 dwt. or 82·2 per cent. of the total contents of the ore treated—an amount that might have been increased to 90 per cent. if the washing of the residue had been more completely carried out.

The liquor remaining after the separation of the gold was next treated with 4·4 lb. of commercial zinc. The black mud precipitated, after standing 24 hours, when washed, dried, and melted, yielded 19·3 dwt. of crude tellurium, or about 0·43 per cent. of the weight of the ore operated upon. The consumption of zinc was about 3 per cent. of the weight of the ore. The argentiferous residues were found to contain 2·5 dwt. of gold and 10·9 dwt. of silver. The final result, therefore, gave about 2 per cent. of gold in excess of that indicated by assay, while the loss of silver was about 8·9 per cent. These differences, especially that of the gold, may be ascribed partly to the difficulty of sampling, owing to the unequal distribution of very rich minerals in the mass of earthy substances forming the ore, and partly to the irregular loss by volatilisation of the precious metals with the tellurium in the assaying processes, which is always observed with those minerals. This method is likely to be of considerable value to the Nagyág and Offenbánya mines, in the event of a demand for tellurium arising on a large scale.

ZINC.—If gold is exposed at a dull-red heat to the vapour of zinc, a brittle alloy is formed, the zinc being taken up by the gold in the same manner as arsenic. This alloy is easily amalgamated by mercury, but of course reduces the standard value of the gold.

Auriferous ores containing zinc are generally also plumbiferous, and are treated as described under Lead.

YIELD FROM PYRITES.—In Victoria, up to the end of 1887, some 119,363 tons of pyrites concentrated from quartz tailings had been treated, yielding 252,240 oz. gold, or an average of 2 oz. 2 dwt. 4·70 gr. per ton. The 4541 tons treated in 1887 gave 9987 oz., or 2 oz. 3 dwt. 23·66 gr. per ton. The highest yield in that year was 6 oz. 12 dwt. 21·06 gr. per ton from 25 tons in Gippsland; and the lowest was 23·32 gr. per ton from 500 tons in Ararat.

CHAPTER VIII.

TREATING AMALGAM, AND REFINING BULLION.

EXCEPT where the gold is collected as a salt or in a pure metallic state, the final operation is to disengage the mercury from the amalgam for further use, and to leave the gold in a separate state, either to be refined or to be sold to the refiner.

TREATING AMALGAM.

In large establishments, the amalgam as collected, whether from sluices when working alluvial deposits or from plates and riffles when dealing with veinstuff, is carried into an apartment containing a strong table, slightly inclined, with a grooved surface near the lower end, and a hole at the lowest point opening into a receptacle for catching any mercury that may escape. On this table is an iron kettle, large enough to hold all the amalgam collected in a single "clean-up." Into the collected amalgam and mercury, some sodium amalgam—1 oz. to each flask of mercury—is introduced, and the whole is stirred. This is considered of great advantage in some places, while in others it is never done. After some time, water is poured into the kettle above the mercury, and the whole is again stirred. Sand and mud rise to the surface, and are removed with a large common sponge. Successive washing and stirring are continued until the surface is comparatively clean. The mercury is then made as dry as possible with the sponge, and the whole is again stirred with the hands. Some dirt generally rises; this is scraped off with a dry card or piece of leather held edge-wise and drawn gently over the surface. This is repeated with occasional stirring, until the mercury seems to be free from mechanical impurity. It is then poured into conical bags of canvas or chamois leather, through the pores of which the mercury runs, or is squeezed, leaving the gold amalgam in the bag. A certain quantity of gold is retained by the mercury, which can only be recovered by retorting; but it is not desirable to do this if the mercury is to be used for further amalgamation, as the presence of gold increases its activity.

The sodium amalgam used in the clean-up must be freshly made to be effective. If kept long, it becomes oxidised and does not give satisfactory results, and the coal oil poured into the bottle by the

manufacturer is detrimental to collection of gold. For these reasons it is sometimes unjustly condemned. It should be prepared by the miner in quantities to suit, and immediately used. The preparation is simple and requires no skill. Metallic sodium may be kept in a wide-mouthed bottle, covered with naphtha or coal oil. It is best to keep it in large pieces, and to cut it into small cubes when required. Enough for a single clean-up can be made in a small frying-pan. The details of the operation are as follows :—

The pan must be perfectly clean and dry. A small quantity of mercury from a fresh flask, or at least not contaminated with mineral impurities, say 5–6 lb., is poured into the pan, and dried, first with a sponge, and then by heating beyond water-boiling-point, but not so as to volatilise any of the mercury. Meantime, a piece of sodium is taken from the bottle, wiped with a perfectly dry rag, and cut into $\frac{1}{2}$ -inch cubes with a common knife, the metal cutting like wax. The mercury is taken out into the open air. The operator places himself on the windward side of the pan, and lifting one of the small cubes of sodium with a long pair of cupel tongs, or pointed wire, places it in the centre of the warm mercury. A flash will follow, and a small quantity of the mercury will be volatilised. Another cube is laid in the same place, and another flash will follow, less intense. This is repeated 3 or 4 times, when the sodium will sink down quietly. At the proper moment, it will be observed that there is a solid mass of amalgam in the centre. It will then be perfectly safe to stir the contents of the pan, when a few more cubes added will change the whole to a mass of beautifully crystallised sodium amalgam, which may be put into closely-stoppered glass bottles, in which it will keep for some time without coal oil or other protection, but when the bottle is opened the whole should be used, as it will soon, otherwise, decompose and spoil. The bottles in which it is kept should be selected to hold just what is required for use at one time. It will be found in every way advantageous to the miner to make his sodium amalgam according to these directions. The precautions to be observed are to avoid the fumes of mercury during the operation, and to be sure that everything used is perfectly dry. After the sodium amalgam is made it can be used with perfect safety.

When taken from the battery of a quartz mill and copper plates, gold amalgam is mixed with sand, metallic iron, bits of quartz, and other mechanical impurities. It is generally gathered in miners' pans or wooden buckets; when the clean-up is large, it occupies a number of vessels, which contain, also, a large quantity of water. This latter is removed by a large sponge, and the whole amalgam is transferred to a single capacious receptacle. From this, any visible lumps of quartz, or other foreign matter, are carefully removed by hand, and a considerable

portion of clean mercury is poured in, the whole being thoroughly mixed. In this condition, all foreign substances float upon the surface, like bits of wood on water. When the coarse fragments of iron or quartz are removed, and the mercury seems homogeneous, the sponge is again used, to remove all water. A piece of soft unglazed paper is then plunged through the mercury, which absorbs any remaining moisture, leaving the mercury dry. A small piece of blanket or flannel, 5 or 6 inches square, or a card, is then held by two corners, and drawn toward the operator, while it is lying flat on the surface of the mercury. If a card is used, it is held nearly vertical. This will clear it of the iron, which, by a dexterous turn of the cloth, may be rejected. The hands are then plunged in, and, by agitation, more iron and sand come to the surface, and are taken off as before. This operation is repeated until the mercury is perfectly liquid and clean, and until no more impurity rises to the surface when freshly agitated.

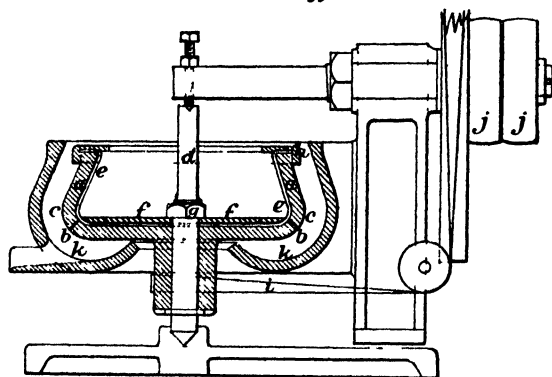
The mercury is then poured into a linen or chamois leather bag of the proper texture, which retains most of the amalgam. When sufficiently drained, it may be removed from the bag. To make the cleaning perfect, the amalgam is squeezed in a piece of chamois leather, or cotton rag, and the hard ball is placed in a clean iron miners' pan and broken down with the thumb. By this treatment it becomes soft, and any fragments of iron or quartz left accidentally in the amalgam become disengaged and may be removed. The whole of the amalgam is again squeezed and the process of "breaking down" with the thumb is repeated until it becomes homogeneous and hard. It is then ready for the retort.

Newbery and Odling have introduced a simple machine (Fig. 259) for separating mercury from amalgam, which possesses advantages over the ordinary method of squeezing amalgam by hand in a chamois bag. The machine consists of a steel vessel *a*, pierced with small holes *b* at its greatest diameter *c*, revolving on a vertical spindle *d*. A chamois leather *e* is stretched over an iron washer *f*, secured by a nut *g* inside the vessel, and is brought up over the top, and there held by the iron ring *h*. When the vessel is rotated by means of the belt *i* and pulley *j*, and amalgam is poured in, the mercury escapes through the leather by virtue of centrifugal force imparted to it by rotation, leaving the amalgam behind; when sufficient has accumulated, the machine is stopped, and the amalgam is removed. The machine is provided with a cast-iron outer vessel *k*, into which the mercury escapes. The operation can be quickly performed, and will, it is believed, effect a large saving in time and also wear and tear of leather. The cost of the apparatus is about 25/.

For small operations the retort used is a deep cast-iron vessel, shaped somewhat like a bowl. The top edge is planed level, and upon this fits a

cover, also planed true, so that when put together the two parts form a perfect joint. From the cover an iron tube rises and bends downward, at an angle of about 20° from the horizontal. The cover is fastened by a clamp and set screw. A mixture of wood ashes and clay is prepared

FIG. 259.



AMALGAM SEPARATOR.

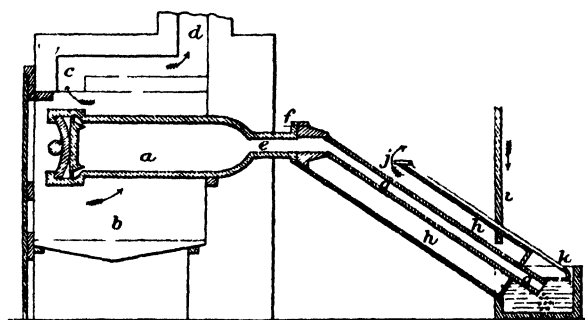
by mixing them into a thick paste with water. When all is ready, the balls of amalgam are placed in the bowl, the mixture of ashes is put thickly around the edge, the cover is fitted, clamp is adjusted, and the whole is firmly fixed by means of the set screw. All the superfluous luting is removed, and the retort is placed in a furnace over a moderate fire. The end of the pipe must dip just below the surface of water placed in any convenient vessel; if the fire is kept well under control there will be no necessity of cooling the pipe. It sometimes happens that when the amalgam has been imperfectly cleaned, the gold will stick to the retort; this may be obviated by chalking the interior of the retort, or putting a piece of common writing paper under the amalgam balls. However, when the amalgam is clean and has been thoroughly worked over, as above described, the gold will come out easily. A very convenient way to retort is to drive two small stakes into the ground, and to fasten a small iron rod to each, at a convenient height; upon this the retort is hung, and around it a fire of small wood is built. When the retort has attained a dull red heat, and no more mercury distils over, the fire is put out and the retort is allowed to cool; the cover is taken off and the bullion is removed. If the amalgam has been properly cleaned, it will be found after retorting to be metallic in appearance, and of a gold colour. It is ready for the melting pot as soon as taken out.

It is never safe to open the retort before it is cool, nor will it stand being cooled in water. Many persons have done themselves great injury in their impatience to see the result of an important run, by opening the

hot retort and inhaling the poisonous mercurial fumes. In extensive runs, when the clean-up is large, a retort of cast iron, made something like a gas-house retort, with movable front door, is set in brick work and furnished with a cooler surrounded by constantly changing water. When it is required to retort mercury, the retort is used in the same way as with amalgam, with this exception: a handful of nails are placed on the surface of the mercury, which prevents violent ebullition of the metal.

There should be a small room expressly for the amalgamator, where the amalgam may be taken for treatment. It should be furnished with a strong wooden trough filled with water. Just above the water a shelf should be placed, upon which to set the pan while cleaning the amalgam. Any mercury accidentally spilled while washing may be removed at any time by drawing off the water. The trough should be furnished with a hydrant, which should give an abundant supply of water. Inexperienced miners find it difficult to separate amalgam from iron, and it is too often put into the retort without proper cleaning, in which case the bullion comes from the retort looking like soot; yet the management of amalgam is simple, and when once understood there need be no failure. Many experiments have been made to clean improperly retorted gold bullion by the use of acids. This is nearly always attended with loss of gold, or is inoperative. Some miners wash the amalgam as taken from the bags, first in diluted sulphuric acid, and then with nitric acid supposed to be pure. Such treatment shows gross ignorance. It is better to properly clean the amalgam and to flux off any accidental impurity, in the crucible.

FIG. 260.



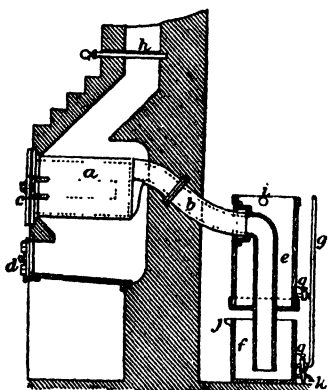
AMALGAM RETORT.

For heating the amalgam, retorts *a* (Fig. 260) are built into fire-brick furnaces *b*, care being taken to construct the fireplaces so as to allow the flames to have full play upon the retorts. The latter are of cast iron,

cylindrical in form, to permit their being turned round should one side become burnt. The usual size at large works is 5 ft. long, the inside cylindrical portion being 3 ft. long by 1 ft. in diameter, and made of $1\frac{1}{2}$ -in. cast iron. The neck *e* gradually contracts to $2\frac{1}{2}$ in. diameter for a length of 2 ft., the end of the neck being furnished with a flange *f*, to which the condensing-pipe *g* is bolted. The condensing-pipe bends downwards, and passes through a body of water contained in a vessel *h*, constructed of boiler-iron, the water supply *i*, and escape *j*, being continuous, and thus ensuring constant coolness, so as to effect the condensation of the volatilised mercury. The amalgam is placed in the retort in cast iron trays fitting the cylinder. *c* is a flue; *d*, chimney stack; *k*, hot water discharge.

In Fig. 261, another form of amalgam retort is shown. *a* is the retort proper; *b*, condensing pipes; *c*, retort door; *d*, fire door; *e*, condensing cistern; *f*, mercury cistern; *g*, cold water pipes; *h*, damper; *i*, *j*, warm water discharges; *k*, mercury discharge.

FIG. 261.



AMALGAM RETORT.

REFINING BULLION.

The following account of the method of refining gold and silver at the New York Assay Office, is taken from E. E. Sage's paper in the 'School of Mines Quarterly':—

There have been and still are a number of different processes by which gold and silver are refined. The one formerly in most general use was the nitric acid process. After the proper proportion of gold and silver are inquarted and granulated, the granulations are treated with nitric acid, the acid dissolving the silver to form nitrate, the gold settling to the bottom of the pots. The solution, after being drawn off, is treated with sodic chloride (common salt dissolved in water); the silver is precipitated as chloride, washed till it is "sweet," and reduced with iron or zinc to metallic silver. It is then refined by fluxing. This process was used in the United States Assay Office till 1866, when Mason introduced the present system, which consists of seven operations:—

(1) *Inquartation and Granulation of the Bullion.*—The gold and silver are sent from all parts of the country, consisting of bullion, amalgams, plate, etc. These are melted and assayed, and are then turned over to the melter and refiner. The bullion is again melted in a large graphite

crucible, the charge being made up so that the gold shall not exceed the $\frac{1}{3}$, nor the copper $\frac{1}{2}$ part of the entire weight. When the metal is melted and thoroughly mixed, it is granulated by pouring, with a peculiar rotary motion, into a cylindrical iron tank, 3 ft. high by 3 ft. diameter, filled with cold water, and supplied with a copper colander, which, when the charge is all granulated, is raised by means of an iron crane, the water nearly all running off. It is then poured on an inclined iron table to allow the rest of the water to drain off.

(2) *Solution of the Granulations.*—The pots used for parting the alloy are made of cast iron, $3\frac{1}{2}$ ft. and 2 ft. in diameter, by 3 ft. and 2 ft. in depth, and are set on brick fireplaces. There are 4 pots holding 160 gal., and 3 holding 50 gal. Each pot is covered with a funnel-shaped hood of sheet lead, with an opening 30 in. long, through which the charge is introduced, and which has a sliding door. From each of these hoods a flue leads to the 2 condensing chambers on the floor above; from the last of these a long winding leaden flue leads into the stack. Both chambers have platforms on which coke is laid several feet in thickness (through which the fumes pass, and are condensed), and are supplied with water sprinklers, so that a spray of water is continually cooling the atmosphere. Nitrous fumes are furnished by soda nitrate, acted upon by sulphuric acid.

About 300 lb. of the granulations are put in each of the larger pots, with about 650 lb. sulphuric acid, and 250 lb. more of the acid is gradually added during 3 hours' boiling. The resulting solution is then siphoned off into the receiving vats on the floor below. The fire being withdrawn, a partial charge of acid is added, and the gold residue is taken out by means of a perforated iron ladle, placed in two of the smaller pots, and boiled for 6 hours with 3 successive charges of about 300 lb. of acid. It is then placed in the third pot and boiled twice. The acid from the last 5 boilings is poured or run off into reservoirs on the floor, and is used again as a portion for fresh granulations. The gold is then placed in a tub, and washed once with cold and twice with warm water; this water is poured into the washing-tub, from which, on settling, the solution is run into a large tank on the floor below.

(3) *Treatment of the Gold Residue.*—The gold is now emptied into one of the filters, washed thoroughly with warm water till it becomes "sweet," drained, boiled in 2 charges of acid for 4 hours, and again washed and sweetened. When it is desired to make very fine metal this process is repeated. It is then thoroughly drained and caked by a hydraulic press.

(4) *Melting and refining the Gold.*—After the gold residue is pressed into cakes, these are placed in an oven and dried, then taken to the melting room, where they are broken up and placed in a graphite

crucible, fluxed for 3-5 hours, and run into bars varying from 997½ to 998½ thousandths in fineness.

(5) *Reduction of the Silver*.—There are for this operation 4 reducing vats, 2 concentrating vats, 8 crystallising vats, and 3 filters, 2 ft. 9 in. diameter by 19 in. deep, with a space below; 2 tanks on a high platform receive the washings from the gold. The sizes of the reducing, concentrating, and crystallising vats vary, on account of having to accommodate them to a given space. The vats and tanks are made of wood and lined with heavy sheet lead. The reducing and concentrating vats are heated by a system of coils of lead pipe, through which steam passes. These vats are enclosed, and connect by a flue with the stack.

Before the silver sulphate is run into the vats, ingots or bars of copper are placed on end around the sides and bottoms, next to the heating coils. The solution is then run in. A sufficient amount of water, together with the weak solution obtained in washing the gold, is added to reduce the strength of the solution to 15°-25° B., which has been found to facilitate the reduction. The sulphuric acid leaving the silver to attack the copper, the silver is precipitated or deposited as metallic silver. It requires 5-6 hours' boiling to complete the reduction.

The resulting copper solution is run off through a filter into a concentrating vat. The silver remaining on the copper bars is scraped off, and the whole is put into a filter. Copper hoes and shovels are used in the handling of the silver. About 2 hours' washing with warm water is required to "sweeten" the silver. It is then put into the draining boxes, made into cakes by a hydraulic press, and dried in an oven similar to that used for gold. When silver containing little or no gold is operated upon, 350 lb. of granulations are dissolved in one pot. With baser metals, the charge of granulations is reduced, copper requiring much more acid to dissolve it than silver.

(6) *Melting and refining the Silver*.—The cakes are broken up, placed in graphite crucibles, and fluxed with soda nitrate, borax, bone ash, and a little soda ash. These reagents absorb the little base metal which remains, as sulphate, in washing. From 4 to 6 hours' are required for fluxing. The silver is then cast into bars of 998½ to 999½ thousandths fine.

(7) *Treatment of the Sulphate of Copper*.—The solution of copper being run into the concentrating vats, is boiled until it reaches the strength of 40° B., and run into crystallising vats. In several days crystals are formed, and the mother liquor is run off into a large tank on the floor below, and thence into the carboys or tanks of purchasers. The crystals of copper sulphate are taken from the sides and bottom of the vats, drained, dissolved in water, and run back into the crystallising vats at a strength of 32°-34° B. The copper sulphate again crystallises, and

the mother liquor is put back into the concentrating vats. The crystals are then taken from the vats, dried, put into barrels, and sold.

If lead is present to any extent in the bullion, it is removed by cupellation before granulating.

The consumption of copper is greatly economised in the reduction of silver by melting silver deposits free from gold and under 400 fine with the copper used for that purpose. The silver in them is thus obtained pure without a direct parting, the copper alloy taking the place of that amount of the purchased metal that would otherwise be used.

The proceeds of the sale of copper sulphate crystals and the liquor, nearly cover the total expense of the acid and bar copper bought.

All wood after it becomes useless is burnt; old, unservicable crucibles are broken up and ground. The grains obtained from them are melted and treated as before, then put through a rough washing and amalgamation process; the resulting tailings are dried, assayed, barrelled, and sold to sweep smelters.

CHAPTER IX.

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THE ARRANGEMENT OF COMPLETE MILLS AND REDUCTION WORKS,
ILLUSTRATING VARIOUS METHODS OF TREATMENT.

THE preceding chapters have necessarily been in a measure disconnected, because each dealt with a portion only of the treatment undergone by the ore, and moreover discussed that portion in detail and under various aspects. It will, therefore, be instructive and interesting to study the arrangements adopted by a few prominent mines in different parts of the world, thus conveying a clear and consecutive view of the methods employed and the results attained. Such information cannot fail to be useful as a guidance in future undertakings, though, of course, each will have its own peculiar circumstances to be considered. In general terms, the selection of the site for a mill determines the cost of dealing with the ore. Primarily, it must be easy of access to the mine and to the roads by which supplies are received, as well as to water and fuel, and it should be so situated that the ore may be carried through all the processes by the exercise of its own gravity.

The mines to be described are taken in alphabetical order. The statements relating to Californian undertakings are chiefly derived from the Reports of the State Mineralogist; the others have been taken from official publications, or were specially communicated.

Amador Gold Mine.—This mine, formerly the M'Kay, is situated in the Jackson Mining District, California. The vein dips 60° east of north, at an angle of about 60° from horizontal; its average width is 20–40 ft. The mine is opened by a tunnel 100 ft. in length, which attains a vertical depth of 60 ft., and by 3 shafts, 280, 250, and 92 ft. respectively. The vertical depth reached in the mine is about 250 ft. The hanging wall is what has been termed, on the "mother lode," greenstone, but later authorities pronounce it diorite. The foot-wall is the peculiar black slate of the mother lode. The mine yields about 60,000 gal. a day of water, which is removed by a 6-in. Garrett "jack head" pump. The National compressor and the National drill are used, and 600 lb. monthly of Hercules powder, No. 2. About 10 ft. a week are made in the tunnel, at a cost of \$6 (25s.) per foot. The cost of sinking the shaft, $8\frac{1}{2}$ by $4\frac{1}{2}$ ft. in the clear, has been about \$30 (6l. 5s.) per foot; the progress has been 3–9 ft. per week. Slate and quartz was the formation

passed through. All the shafts are timbered, and double compartment from top to bottom. Sawed timber, spruce and pine, is used in the shafts, 8 by 9 in. and 12 by 12 in., at a cost of \$20½ (4*l.* 3*s.* 5*d.*) per thousand. Round timbers, 16 ft. long, cost 75 cents to \$4 (3*s.* 1*d.* to 16*s.* 8*d.*) each, according to the diameter at the smaller end. All the mines along the lode are 16–25 miles from timber. The ore will be conveyed by a tramway, 2200 ft. long, from the mine to the new mill now being built. The ore is to be amalgamated in the battery and on outside plates; the tailings to be passed over Frue concentrators, and the sulphurets are to be worked in the company's chlorination works, which will have a capacity of 3 tons per day. The mill in process of erection is a 60-stamp mill, with 24 Frue concentrators. The stamps will weigh 850 lb. each, and will drop 6 in. 90 times per minute. The duty of each stamp will be 2½ tons in 24 hours. The shoes and dies will be of white iron, costing 4 cents (2*d.*) per lb. The screens will be slot punched, dimensions inside the frame 6 in. by 48 in., and will be set inclined. The size of apron will be 4 by 7 ft., and of the sluice 16 in., with a length of 20 ft. The aprons and sluices will be covered with silvered plate, and will have an inclination of 1½ in. to 1 ft. Hendy Challenge feeders, a clean-up pan, and revolving barrel will be used. The cost of working sulphurets is estimated to not exceed \$13 (2*l.* 14*s.* 2*d.*) per ton, whether worked by the Plattner process or by that used at Bunker Hill, when 3 tons or more per day are worked. Thus:—

Estimate Plattner Process.

	\$	c.	£	s.	d.
Labour	4	00	=	0	16 8
Wood	3	00	=	0	12 6
Peroxide manganese	0	65	=	0	2 9
Salt	0	35	=	0	1 5
Sulphuric acid	2	00	=	0	8 4
Incidentals—taxes, insurance, ordinary repairs, iron sulphate, assaying material, &c.	3	00	=	0	12 6
	<u>\$13</u>	<u>00</u>		<u>£2</u>	<u>14 2</u>

Estimate Bunker Hill Process.

	\$	c.	£	s.	d.
Labour	4	00	=	0	16 8
Wood	3	00	=	0	12 6
Lime chloride	1	20	=	0	5 0
Salt	0	15	=	0	7
Sulphuric acid	1	26	=	0	5 3
Power	0	39	=	0	1 8
Incidentals, &c., same as by Plattner process..	3	00	=	0	12 6
	<u>\$13</u>	<u>00</u>		<u>£2</u>	<u>14 2</u>

The superintendence and office expense is no greater than with the general mining and milling business alone, and therefore does not enter

into the cost here. It is estimated that 90–95 per cent. of the value of this grade of sulphurets can be saved by either process. The number of men employed is 30. The outside work varies, the mill being under construction. The average wages paid in the mine is \$2·62 (10s. 11d.); outside \$2 (8s. 4d.). In the mill the average will be \$2·75 (11s. 5d.). Wood costs per cord (\$5·50 22s. 11d.) on the average, and will not be used for steam purposes, excepting in case of failure of water supply, but will only be used at the chlorination works regularly, and at the rate of about $\frac{1}{2}$ cord a day. The hoisting works will require 125 in. of water, with 165 ft. of pressure; the mill will use 100 in., under 391 ft. of head, besides the water from the hoisting works used over, with a pressure of 226 ft. The cost of water is 20 cents (10d.) per inch for 24 hours. Hoisting works cost per day for water power \$25 (5l. 4s. 2d.); mill power, \$20 (4l. 3s. 4d.); total, \$45 (9l. 7s. 6d.).

Boss Standard Continuous Process.—This was introduced in 1887 by M. P. Boss of San Francisco, and is claimed to be specially adapted to those ores in which the gold is in a very fine state or is “rusty.”

The following account relates to the Jay Gould mine, Helena, Montana. The mill is a 10-stamp. The ore assays 1·247 oz. gold and 5·42 oz. silver per ton of 2000 lb. The gold is extremely finely divided. The ore is a close-grained quartz carrying some manganese oxide and iron pyrites. As the ore is delivered at the mill it is more or less mixed with a hard argillaceous rock and a soft clay, both of which produce abundant slimes under the stamps. The process is essentially as follows. The ore is crushed in an ordinary battery fitted with No. 3 slotted screens, barely enough water being used to run out the pulp. From the battery, the pulp is discharged through a $2\frac{1}{2}$ -in. pipe into the centre of a small grinding pan, $3\frac{1}{2}$ ft. inside diameter. The muller shoes and pan-dies are solid circular rings with radial grooves for facilitating the discharge of the ground pulp, after the manner of the old-fashioned stones in flour mills. To ensure grinding, and compensate for the wear of the shoes and dies, a stout steel spring is fitted under the cap of the driver on the pan-spindle; a turn of the wheel from time to time sets down the spring. The bottom of the grinding pan is on a level with the top of the main line of pans. The muller runs at 72 revolutions a minute. From the grinding pan the pulp discharges into No. 1 of the main line of pans (of which there are 6). Pans No. 2 and 5 are fitted with automatic arrangements for introducing chemicals. At first salt and bluestone were introduced into the charge at pan No. 2, and quicklime at No. 5. Experiments proved that the use of bluestone increased the percentage of silver saved, but diminished that of the gold. The use of salt alone had no effect, hence the use of these chemicals was discontinued. The alkaline action of the lime was found beneficial for keeping the mercury

clean and reducing any "floured" mercury. In running, the muller in pan No. 1 is set to grind, while the remainder of the pans run with the mullers up, the rate being 72 revolutions a minute. The pans (see p. 494) are heated by exhaust steam from the engine. About 200 lb. fresh mercury is introduced into each pan except No. 1, which is only charged once on a run, every hour, the same amount having been withdrawn previously. The charging and discharging of mercury is effected by a siphon tap at the bottom of one side of the pan. The discharged mercury runs through pipes to the amalgam safe. The strained mercury from the safe is raised by an elevator to a tank above the level of the pans, and by a line of pipes is drawn into the siphon, or more properly speaking, mercury trap, when needed for charging. About 4 in. from the top, the pans are connected by 4-in. piping. In the same manner, the pan system is connected with the settlers, these latter being similarly connected with each other. Through these communications, the pulp flows continuously, taking about 8 hours to pass through the system. In the settlers, of which there are 3, the pulp is diluted by means of a stream of fresh water. The settlers run at 16 revolutions a minute. Any pan can at any time be cut out of the system, the pulp being made to pass it by means of an arrangement on the injector principle. The pans are emptied by a steam siphon.

Bunker Hill Gold Mine.—This lies in the Amador Mining District, California. The vein runs N. 15° W., and dips 15° north of east at about 75° . Its width varies from 3 to 25 ft. There is an adit level and 2 incline shafts 350 ft. apart. The north shaft is 800 ft. deep on the incline and 345 ft. vertical. The adit level is 1100 ft. long, reaching a depth of 300 ft. vertical. The formation of the hanging-wall is diorite (greenstone). The foot-wall is slate. The water is removed from the mine with a Hooker steam pump, running 3 hours a day, at the rate of 60 strokes a minute. Hercules, No. 2, powder is used, and 1500 lb. are consumed monthly. The cost of mining, per ton, is about \$3 35 (13s. 11d.). The cost of running the adit level, for 1100 ft., was \$3 50 (14s. 7d.) for labour, under contract. The tunnel passes through black slate the entire distance run, 4 men making 20 ft. per week. The formation passed through in the shafts was black slate. The shafts and tunnels are timbered, sawed pine and spruce being used in the shafts. All other parts of the mine are round timbered. Sawed timber has an average yearly cost of \$22 50 (4l. 13s. 9d.) per 1000 feet. The timber is situated 18-20 miles from the mine. The ore is hauled from the mine to the mill, over a tramway; cost of transportation is 5 cents (2½d.) per ton. The ore is the common ore of the mother lode in Amador County, being quartz and black slate mixed, and free milling, with the exception of 2 per cent. of sulphurets, which are saved by

concentration. The ore is crushed in a Hendy crusher 9 by 14-in. ; pulverised under stamps, amalgamated in the battery and on outside plates. Concentrations are saved on Frue vanners, and are reduced at the company's works. The tailings are afterwards let out on contract and concentrated on canvas. The mill is arranged for either water or steam power. A 4-foot Knight wheel, under 260 ft. pressure, drives the Hendy crusher, and a 6-foot Knight wheel, under 270 ft. pressure, drives the mill. In case of failure of water for power, the mill is driven by an engine with a 17-inch cylinder and $3\frac{1}{2}$ -foot stroke, connected with steam boiler 54 in. by 16 ft. long. The pattern of motor is Knight. The discharge is 7 in. above the new die. The order of drop of stamps is 1, 5, 2, 4, 3. The number of tons, on the average, crushed, when a No. 8 screen is used, is 99 a day through the month, including stoppages. A trial is being made of steel shoes and dies, to compare with those of iron. The cost of steel is $9\frac{1}{2}$ cents ($4\frac{3}{4}d.$), of iron 4 cents ($2d.$), with 2 cents ($1d.$) per lb. returned for old iron. Shoes and dies weighing 8800 lb. will crush 4700 tons of Bunker Hill ore before becoming thin enough to be replaced. The old iron remaining will weigh about one fourth of the original weight. Therefore the cost of iron per ton crushed is $6\frac{1}{2}$ cents ($3\frac{1}{4}d.$) The water used in the battery and concentrators is 8 measured inches, under 4-inch pressure above the top of the opening. The screens $\frac{1}{4}$ -inch angle slot, last 20 days on the average, making the cost of a ton of ore worked, for screens used, about 1·2 cent. The dimensions of the screen inside frame are 46 in. by 5 in. ; they are slightly inclined. Brass wire screens, thoroughly tested, were found very unprofitable : a proper quantity of ore could not be passed through the screen, owing to clogging. Besides requiring extra care and attention, the results were otherwise unsatisfactory. The percentage of recovery of gold in the battery (including the splashboards outside), is 90 ; on outside plates it is 10 ; the loss of quicksilver is 900 oz. monthly ; absolute record, covering a period of 6 months, is 0·3 oz. per ton, or a little over 1 cent ($\frac{1}{2}d.$) per ton of ore crushed. Frue concentrators (16) are used for saving the sulphurets, with lateral motion, at the usual speed of 200 per minute. The sulphurets are pyrites, with trace of sulphurets of antimony, arsenic, and lead, like all other sulphurets of the mother lode. The sulphurets are roasted with 1 per cent. salt, in a furnace which is a combination of a reverberatory and horizontally revolving roaster. The roasted ore is subsequently treated in a lead-lined revolving chlorination barrel, in which the chlorine is generated from lime chloride by the action of sulphuric acid. Two cords of pine wood are used per day for steam purposes, and $1\frac{1}{4}$ cords for roasting, at a cost of \$6 (25s.) per cord. The hoisting works are run by steam. The north shaft is provided with a double 8-inch cylinder engine, and connected with pinion and spur-gear.

A splashboard is used on the outside of the battery, 8 in. by 46 in. in size, covered with silvered plate; the size of the apron is $3\frac{1}{2}$ by 4 ft., and the width of the sluice is 14 in. by 14 ft. in length. The plate on the inside of the battery is 2 in. on the perpendicular side, and $4\frac{1}{2}$ in. on the incline side. A strip is introduced underneath to raise the plate, and is removed as the dies wear them. The inclination of the apron is $1\frac{1}{2}$ in. to 1 ft.; the sluice is inclined $1\frac{7}{8}$ in. to 1 ft. Six Templeton, or roller, and 2 Hendy challenge feeders are used. At the south hoisting works a 10-inch cylinder single engine is used. The cost of water is 20 cents (10*d.*) per inch; 105 in. in all are used: 8 miners' inches run the rock breaker under a head of 260 ft.; this water afterwards supplies the batteries and concentrators; 92 in. supplies power to the mill, under 270 ft. head, and 5 in. supplies the chlorination and outside works. The milling at the Bunker Hill may be segregated as regards cost, as follows: Shoes and dies, $6\frac{1}{2}$ cents; screens, $1\frac{1}{2}$ cents; quicksilver, $1\frac{1}{2}$ cents; labour, 17 cents; water, 20 cents; incidentals, 14 cents; total, 60 cents (2*s.* 6*d.*).

The chlorination process in use at this mine is nowhere else employed on this coast; the cost has been found to be no greater than the Plattner process, and it is claimed for it a recovery of a very much higher percentage of gold. When the sulphurets from the Bunker Hill are worked by this process but \$3 50 (14*s.* 7*d.*) per ton remain in the tailings, whereas, when the Plattner process was used on sulphurets of equal assay value, they were found to contain about \$7 (29*s.* 2*d.*) per ton. Sulphurets containing \$57 (11*l.* 17*s.* 6*d.*) per ton are worked up to 92 per cent. of the assay value by this method, and tailings from \$100 (20*l.* 10*s.* 8*d.*) sulphurets do not contain any more gold than those of a lower grade. The roasting furnace proper is 9 ft. wide and 40 ft. long, outside measurement; the thickness of walls is $1\frac{1}{2}$ ft. The top of the bridge wall is 18 in. above the grate bars, and 9 in. below the centre of the furnace arch; the drop to the hearth is 6 in. That part of the furnace nearest the bridge wall is a finishing oven, with revolving hearth, 12 ft. diameter, inside measurement, and makes 1 revolution per minute. The furnace is charged in the same manner as ordinary reverberatory furnaces for roasting sulphurets, and at this mine 1 per cent. of salt is introduced with the charge. During the latter part of the roasting in the finishing oven, different portions of the sulphurets are in succession exposed to the highest action of the heat and oxidising flame. The capacity of the furnace is 2 tons in 24 hours; the fuel required is $\frac{3}{4}$ cord per ton of sulphurets; the wood employed is pine, and costs \$6 (25*s.*) per cord. Water for power to revolve the hearth and barrel costs 50 cents (2*s.*) for each ton of sulphurets worked (5 miners' inches, under 260 ft. head, at 20 cents (10*d.*) per inch a day). The barrel used for chlorinating the roasted sulphurets is,

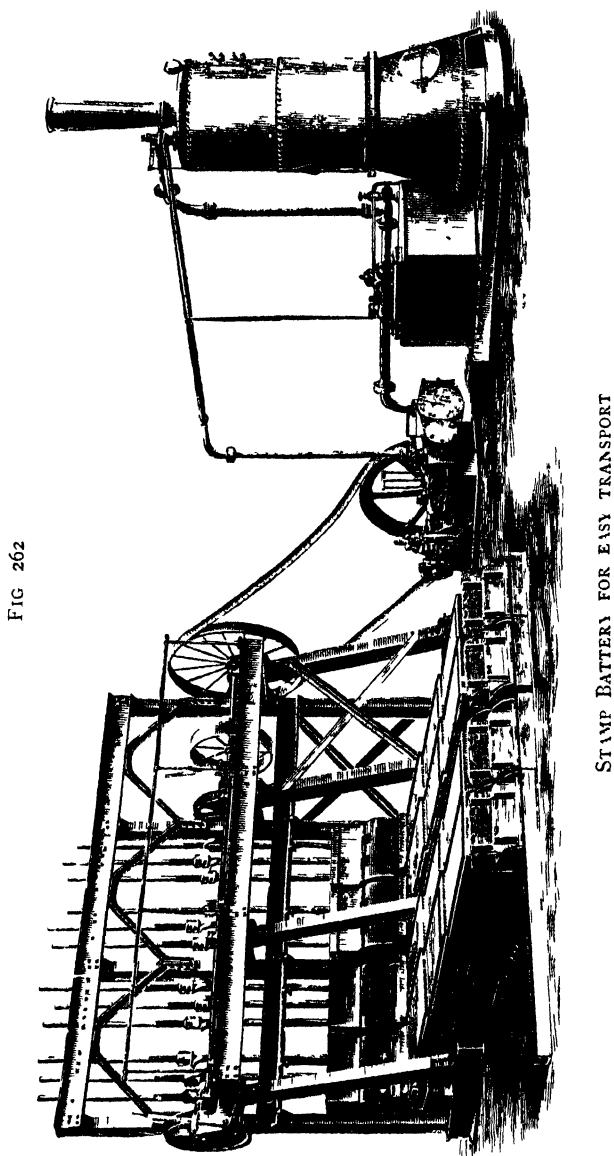
in outside diameter, 40 in ; in outside length, 54 in. ; inside length, 48 in. There is an opening in the centre of the barrel, 10 in. diameter, used for the purpose of admitting and removing the charge. The barrel is lined with $\frac{1}{4}$ in. lead of the best quality, and entirely free from zinc. After the charge has been introduced and the cover of the opening is closed, the barrel is made to revolve 13 times per minute for 6 hours, when the mass is discharged into a leaching tank of the usual construction. The only skilled labour required in this process is in the roasting. Two tons are worked per day, the barrel being charged twice in the daytime, with one ton at a charge. The leachings from the roasted pulp are treated as in the Plattner process. In charging, 130 gal. water are first introduced into the barrel ; 30 lb. dry lime chloride, after being mixed on the floor with 1 ton of roasted sulphurets, are next put in ; 36 lb. sulphuric acid at 66° B. is then added, and the cover is fastened as soon as possible.

Cost of Working.

	£	s	d
Labour, per ton	4	75	= 0 19 9
Power and water	0	50	= 0 2 1
Wood, $\frac{1}{8}$ cord at \$6	3	75	= 0 15 8
30 lb. lime chloride, at 4 cents	1	20	= 0 5 0
36 lb. sulphuric acid, 66°, at 3½ cents	1	26	= 0 5 3
20 lb. salt, at $\frac{1}{4}$ cent	0	15	= 0 0 8
General expenses—assaying, melting, taxes, insurance, repairing, loss of material in handling, interest, &c.	3	00	= 0 12 6
Total	\$14	61	£3 0 11

Calcedonian Battery, Maldon, Australia.—The quartz is first burnt in kilns, of which there are 5, each capable of holding about 50 tons. The time occupied in doing this is 9 days. The calcined quartz is trucked from the kilns to hoppers, and is hand-fed into the stamper-boxes, about 3 oz. or less of mercury being put into the boxes every 3 hours, according to the richness of the quartz operated on. The crushed quartz must be fine enough to go through gratings 200 holes to the sq. inch. The sand and water then flows into an iron ripple, charged with 25 lb. of mercury, and from there it flows into a Chilian mill, also charged with 25 lb. of mercury. Each mill receives the discharge of 4 stamps. From the mill, the discharge flows over 6 ft. of copper plates into a shoot that distributes the sand and water over blanket-tables 15 ft. long. These blankets are washed every hour, and the sand and mineral saved are put through a large rocking cradle lined with baize. The material saved in the large cradle is put through a small one, and the stuff from the small cradle is put into a camp oven with caustic soda, water, and mercury, and boiled for 5 hours. It is then washed off in the usual manner,

and the mineral left is burned in a reverberatory furnace, and then ground in Chilian mills. After the sand has passed over the blanket-tables it flows into a pit, and is elevated by a sand pump, and discharged into an iron tank, which distributes it over 3 sets of strakes



150 ft. long and 16 in. wide each. These are skimmed every 2 hours, and the material collected is ground in Chilian mills with mercury.

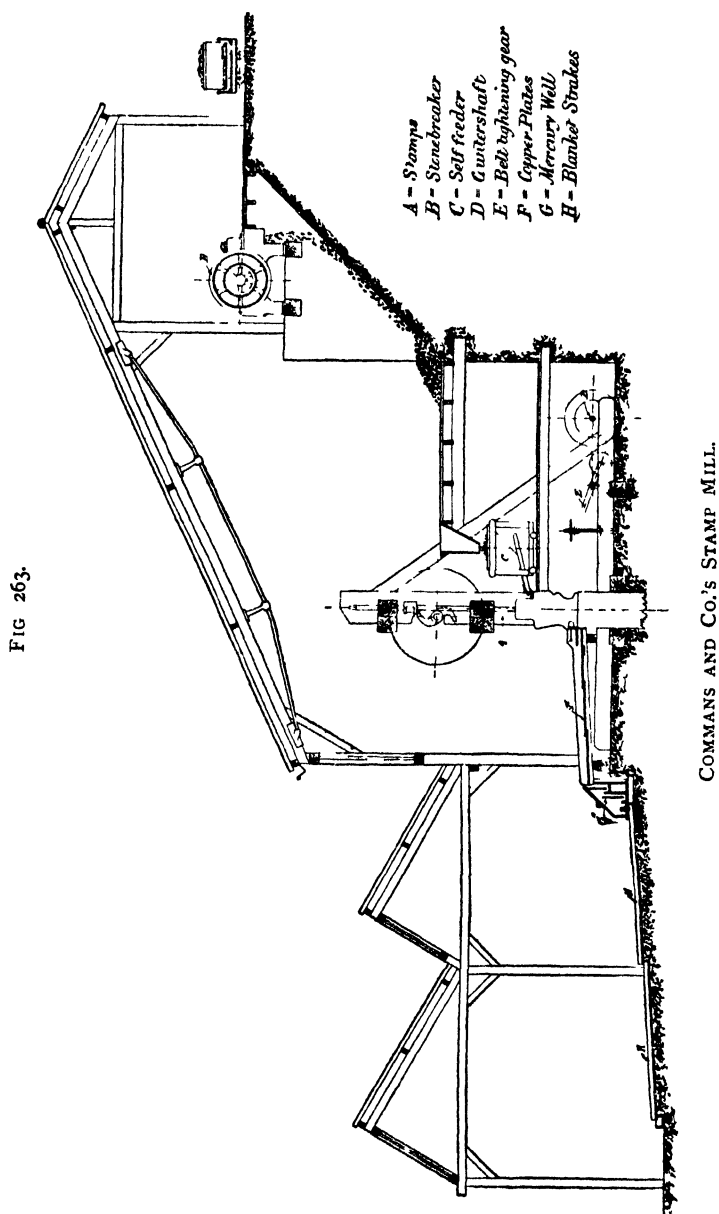
Calvert, Cornes, & Harris's Mill.—Fig. 262 illustrates a stamp

battery of 12 heads, constructed with steel framing. It was designed to accommodate itself to difficulties of transport, hence the limit of weight for any piece was 20 cwt. For this reason the cast steel mortars only receive 3 stamps each. The mill is driven by a 20-H.P. nominal compound engine, made specially light, in wrought iron frame. A Cornes patent vertical sectional boiler provides steam. The whole was built by Calvert, Cornes, and Harris, 30, Walbrook, London, for the Essaman Gold Mining Co.

Commans and Co.'s Mill.—Fig. 263 shows a mill largely adopted for the treatment of "free milling" gold ore, built by Commans and Co., 52, Gracechurch Street, London. The ore, as it comes from the mine, is either tipped from the trucks alongside the stonebreaker and shovelled into the self-feeders; or, if the site will allow of the necessary fall, is tipped over an inclined screen of iron bars, through which the smaller pieces fall directly into the ore feeder bin. The larger pieces, sliding down over the screen, are reduced to a suitable size in a stone breaker. The finely-broken ore in the self-feeders is evenly and automatically delivered into the mortar boxes, water being introduced at the same time. The falling stamps crush the ore into a pulp, which, when sufficiently fine, passes out through perforated screens in the sides of the mortar boxes, and flows in a thin film over gently inclined tables covered with amalgamated copper plates, the pulp in its passage over these plates yielding up a large portion of the fine gold it contains. Often amalgamation is begun in the mortar box, into which a certain amount of mercury is introduced, the amalgam being collected on copper strips fixed inside the box. Any small percentage of sulphurets or rusty gold that may be contained in the ore can be caught on the blanket strakes. If the ore contains a large percentage of sulphurets, the tailings should pass over Frue Vanners, of which Commans and Co. hold the sole licence for manufacture in this country.

Haile Gold Mine, Carolina.—The stamps weigh each 750 lb., and the average number of drops is 80 per minute. The fall of the stamps is 7 in. average. The screens used are 40-mesh. The amount of ore put through daily is 35–37 tons. The average value of the ore is \$11–12 (45–50s.) per ton. Of this amount, \$3½–4 (14s. 7d.–16s. 8d.) is free gold, the remainder being combined and more or less enclosed in the sulphides. The mortars are provided with inside plates, which are taken out and cleaned every day, yielding \$75–80 (15l. 12s. 6d.–16l. 13s. 4d.) a day. Each battery is also provided with 10 ft. of copper plates outside, the average catch of which is about two-thirds more than the inside plates. At the bottom of the plate-tables are a series of small spitzkasten, with an upward current of water, which are very effective in catching whatever mercury is lost from the plates. All the tailings are then run over con-

centrators, whereby the sulphurets are saved, the average tailings, which finally leave the mill, showing less than 30 cents (1s. 3d.) of gold per

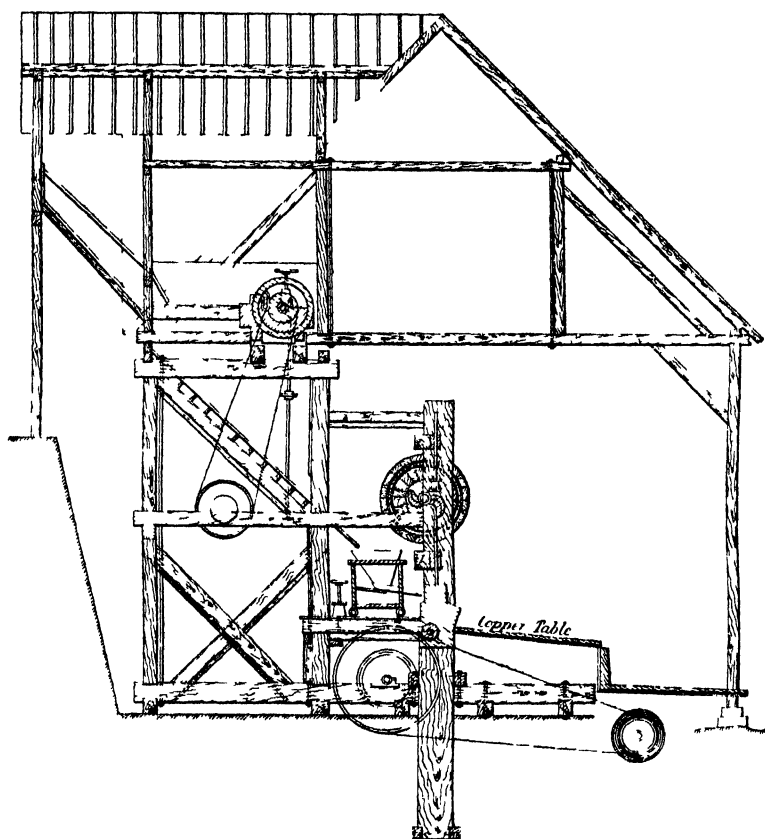


ton. The entire cost of milling, including labour, cordwood, wear and tear of machinery, and labour of concentration, is 51 cents (2s. 2d.) per

ton, to which should be added 2 lb. of shoes and dies to the ton of ore, say 9 cents (4*d.*), making in all a total cost of 60 cents (2*s.* 6*d.*) per ton.

Hite Mine—This mine is located in the Mariposa District, California, at an elevation of 2080 ft. above the sea. The course of the vein is east and west, with a northerly dip of 75° and an average width of 8 ft. The formation of both hanging and foot-walls is slate. The mine is worked through 2 tunnels, the upper being 700 ft. in length, and the lower (which taps the vein 900 ft. on the incline) 1400 ft. From the point where the lower tunnel taps the vein, an incline shaft has been sunk

FIG. 264



HITE MINING CO.'S MILL.

300 ft. in depth. Levels, run from either side of the incline, disclose a large body of ore. An engine run by compressed air is used for hoisting the ore to the tunnel, whence it is transported in cars to the surface, and carried by tramway to the ore house adjoining the mill. The apparatus

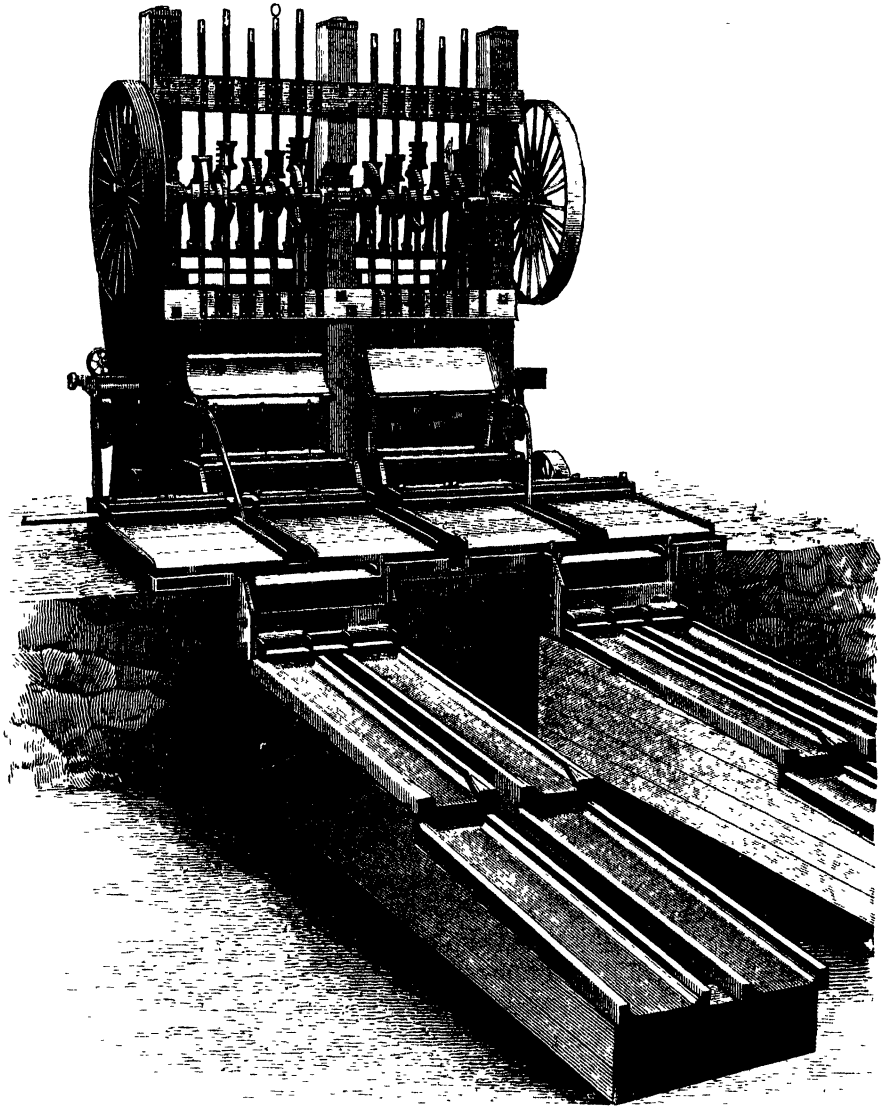
consists of a 40-stamp mill, a pan mill, and an air compressor, all run by water power. The stamps weigh 750–800 lb. each, fall 5–8 in., 80 drops per minute, and crush $1\frac{3}{4}$ ton a day to each stamp. The plates used are an apron 4 ft. wide and 6 ft. long, connecting with a sluice 20 in. in width by 14 ft. in length. The apron has an inclination of $1\frac{1}{4}$ in. to the foot; the sluice 1 in. to the foot. The water used here is conveyed to the mill through a ditch 1 mile in length, furnishing a 46-foot head, which drives a 48-inch Leffel turbine wheel, giving the power necessary for running the 40-stamp mill. This water, after leaving the mill, is picked up and conveyed by flume and ditch, and applied to a 24-inch Leffel turbine wheel, under a 30-foot head, furnishing power to run the pan mill and the air compressor, located about 1 mile from the mine. The air is forced through 3-inch iron pipe to the hoisting engine, located 5 miles from the mine. This company has a steam sawmill, with a daily capacity of 6000 ft. of lumber. There is no separate account kept of the difference between the battery and plate savings, merely a record of the general clean-up. The sulphurets (iron pyrites, zinc blende, and galena), assay value \$80 (16*l.* 13*s.* 4*d.*) per ton, are allowed to escape, no provision having been made for saving them. The ore contains about 2 per cent of sulphurets, for the saving of which the company intend soon to put in concentrators. When Hite, the discoverer, owned and worked the mine, the tailings were taken by car to the pan mill below, and some of them worked over, but with what results could not be learned. There is a large accumulation of these tailings, the pan mill not being run at present. A section of the mill is shown in Fig. 264.

Hornsby & Son's Mill.—Fig. 265 shows the type of Californian gravitation stamp mill that has been very largely constructed for the South African and other gold fields, by Richard Hornsby & Sons, Limited, Grantham. The frame-work of this mill consists of the usual uprights, but has only one diagonal stay at the back of each upright. When massive proportions are adopted, this is found to be the most efficient way of constructing the frames of gravitation stamps, as it gives ample room in front to put in larger launders when they are required. This has been specially the case in the South African gold mines, and the width of launders used there has been made double that which is ordinarily used in other parts of the world. Blanket strakes are shown at the end of the launders. Amalgamating plant is also supplied where it is necessary to use it in the treatment of the different ores that are encountered.

Wood is probably more used than any other material for battery-frames, though Messrs. Hornsby also construct batteries with wrought-iron frames, made of massive channel iron, put together with turned bolts, as rivets have been found objectionable, from the heavy jarring that takes place in batteries, and which causes the rivet heads often to shake off.

In Fig. 265, the ordinary tightening arrangement, which consists of a pulley pressing against the driving belt, is used, but this has been altered in many cases by using a friction clutch, which is attached to a pulley on

FIG. 265.



HORNSBY'S STAMP BATTERY.

the countershaft, at the lower end of the frame. The counter shaft in many cases is placed at the back of the uprights, as that is found most convenient in giving clear room in the front.

Automatic feeders (which are not shown in the illustration) are also arranged, and are a modification of the well-known "Tulloch" feeder (see p. 443) which has proved to be one of most efficient feeders that have been brought out for dealing with the many descriptions of ore.

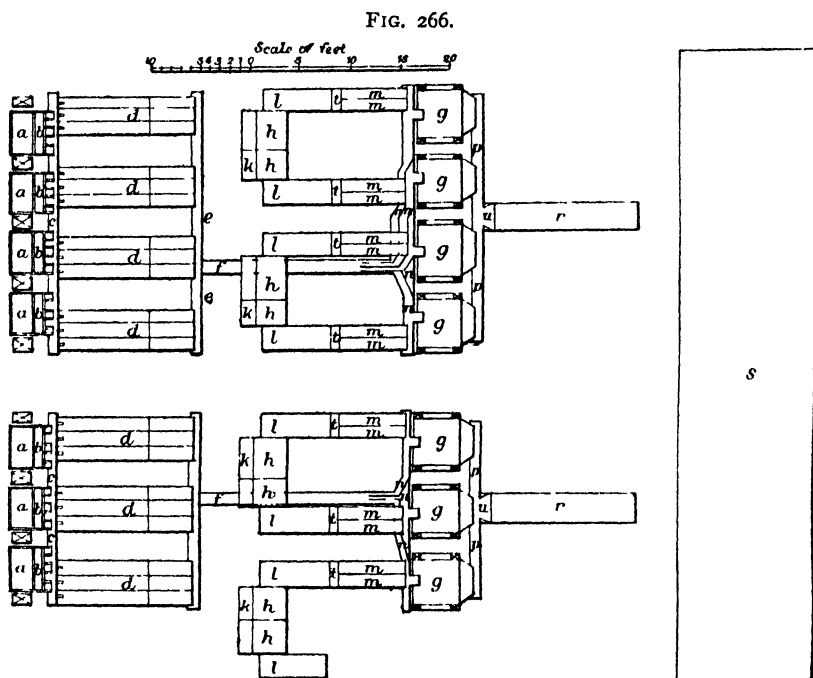
Idaho Mine.—This mine, situated in the Grass Valley District, California, has had a long and successful history, having produced a total of over \$8,000,000 (say 1,700,000*l.*), nearly one-half of which has been disbursed to the shareholders in dividends since 1869. The present total depth of the incline is 2696 ft., vertical depth 1790 ft. The average width of the vein in the lower workings—length of ore shoot 1200 ft.—is given at 30 in. In 1887 were crushed 26,686 tons of rock. The water raised daily by the Cornish plunger pumps amounts to 21,600 cub. ft. The mine employs 173 men, and the mill 17, while 16 are occupied on outside work. The wages paid are \$3 (12*s.* 6*d.*) a day for men in the mine and mill, and \$3-5 (12*s.* 6*d.*-20*s.*) on outside work. Mining and milling per ton of ore cost \$8.44 (1*l.* 15*s.* 2*d.*). The screens have been changed from No. 5 slot to No. 6 slot. The mill, of 35 stamps, uses 36 miners' inches of water (6-inch pressure), and 320 miners' inches are required under a pressure of 542 ft. to drive the Pelton wheels.

The tailings from the mill are being re-treated by Cobb & Norton, by being passed, with additional clean water, through shallow sluices or strakes, lined with canvas, to which the *fine* sulphurets cling and are washed off periodically by means of a jet from a hose and nozzle, and being afterwards collected, they are further cleansed by a repetition of the process. The resulting concentrated material is worth \$30 (6*l.* 5*s.*) per ton and upward.

Below, on the creek, the tailings from Cobb & Norton's works are taken up by Bush & Gauthier, who pass them through sizing boxes, and again over strakes lined with gunny sacks, which are periodically lifted and washed in tanks. The coarse sulphurets lost above being thus recovered, are mixed with sand and treated by grinding and amalgamation in 8 Knox pans, driven by a small overshot wheel. In these pans, besides mercury, nitrate of mercury and sometimes bluestone are used, the operators being of the opinion that they get nearly all the gold. The 8 pans work 1½ tons of stuff in 24 hours. A part of the gold obtained by this latter process comes, no doubt, from the coarse sand, in the grains of which it is enclosed. The nitrate of mercury probably acts by precipitating a film of mercury upon the iron surface of the pan and upon each particle of iron ground off from the shoes and dies. The mercury *in statu nascendi* probably tends, without extra energy, to attach itself to the liberated particles of gold. The bluestone, in all probability, assists in the formation of an amalgamated film on the iron, by coating it with copper; but as it is used only occasionally, and for the purpose

of preventing a large loss of mercury, under certain conditions, its principal use seems to be to dissolve such particles of iron as have become amalgamated through the action of nitrate of mercury, and thus produce a powdery sort of amalgam, which, when the iron is replaced by copper from the bluestone, is converted into a soft, coherent amalgam, easily saved in the washing. The Idaho Mill is provided with an extensive system of strakes, over which the tailings are first passed, after leaving the mill, where they have already flowed over the blankets, so that all the ore is subjected to no less than four concentrations.

A plan of the Idaho Mill is shown in Fig. 266. The lower holes of the screens when new are 3 in. above the discharge, which takes place



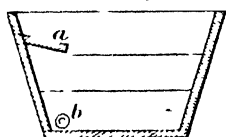
IDAHO MILL, GRASS VALLEY, CALIFORNIA.

from the battery *a* into a splash-box *b* running the whole length of the mortar. This box is 9 in. wide on the bottom. The back is the whole height of the screen; the front is only 6 in. high, and is closed along the front, having 3 openings 6 in. long and 3 in. high, equally dividing the length of the box, discharging into the trough *c*, which distributes the pulp over the blanket sluices *d*, of which there are 3 for each battery of 5 stamps. These sluices are made of $1\frac{1}{4}$ in. planed boards, 16 in. wide. The sides are $3\frac{1}{4}$ in. wide, and are nailed on to the edges of the bottom so as to make the sluice 2 in. deep in the clear. Each sluice is made in

2 parts; the upper is 9 ft. 6 in. long, and the lower 4 ft. 6 in.; between the sluices is a drop of 3 in. Two are worked at a time, while the third is being cleared.

The sluices are so laid that while having a regular pitch of 3° – 4° longitudinally they will be perfectly level transversely. This is necessary to ensure the same amount of flow over all parts of the sluice. At their ends the sluices project about $1\frac{1}{2}$ in. over a trough *e*, 6 in. wide, which connects with another *f* (there being one for each set of 2 batteries on one side of the mill and 4 on the other), which is 18 ft. long and 13 in. wide. These are filled with copper plates and discharge upon a Eureka rubber, which discharges upon a copper plate sluice *r*, 16 ft. long and 32 in. wide. The first sluices *d* are designed to receive the blankets, and have a grade of $\frac{3}{4}$ in. to the foot, and upon them the battery discharges its pulp. The blankets are manufactured for the purpose, and are 6 ft. long and 20 in. wide, and weigh 1 lb. 10 oz. each. They are made of coarse grey wool, and have the nap on one side only; the other side, which is placed against the bottom board of the sluice, is carefully shorn. The upper sluice receives 2 of these blankets which overlap each other, the lowest being placed first. The lower sluice has but 1 blanket. The object of the blanket is to arrest the fine particles of gold and sulphurets, which are caught in the fine fibres of the nap. The blankets wear out at the rate of $1\frac{1}{2}$ yd. per stamp per month; they cost $\$1\frac{1}{2}$ – $\$2$, (6s. 3d.–8s. 4d.) per yd. at the mill. At the end of these blanket sluices there is a passage 30 in. in width, and in front are 2 tubs *h*, made of $1\frac{1}{2}$ in. boards, 3 ft. 6 in. wide at the top and 2 ft. 6 in. at the bottom, and 30 in. deep, one 3 ft. and the other 4 ft. long. They are provided with a series of plug holes *b*, for discharging the water gradually, and each has a shelf *k* (Fig. 266) or *a* (Fig. 267), running the whole length of the box, for convenience in washing the blankets.

FIG. 267.



WASHING TUB.

The blanket washings, when settled, are passed into Attwood amalgamators. The sands from thence go into a box *l*, 1 ft. deep and 21 in. wide, which feeds 2 amalgamated copper plate sluices *m*, 8 ft. long, 21 in. wide, with a grade of 1 in. per ft., whence they discharge into Eureka rubbers *g*. These discharge over aprons, the sides of which converge to an opening 27 in. in width, and into a trough 6 in. wide, leading to a sluice *n*, made of 1 in. plank 15 in. long and 32 in. wide, from which there is a drop of 3 in., where it discharges into another sluice *r* of the same width, but $16\frac{1}{2}$ ft. long, with a grade of $\frac{1}{2}$ in. to the foot. The sides of this sluice are 3 in. high, and the sluice contains amalgamated plates 30 in. wide. From here the tailings are discharged into settling vats to be concentrated. A certain amount of skimmings,

which varies between $\frac{1}{2}$ and $\frac{1}{10}$ per cent. of the battery sands, are taken from the amalgamators, and the riffles which are placed in the sluices in front of them. The skimmings are ground and amalgamated in a Knox pan.

Keystone Consolidated Mining Company.—This company's property is situated in the Amador Mining District, California. The ores have been mined and milled constantly for 25 years. The course of the three parallel veins is slightly west of north, and east of south. The dip is a little north of east; the company mines over a space of 400 ft. in width, in which the parallel veins occur, varying in thickness from stringers to 60 ft. in width, and in dip from 45° to 60° from the horizon. The property is equipped with two shafts; the depth of the main shaft is 1305 ft. on the incline, and reaching a vertical depth in the mine of 1000 ft.; the south shaft is used as an air and water shaft, and reaches 1024 ft. in depth on the incline, or 780 ft. perpendicular. The hanging-wall is diorite; the foot-wall has not been reached. The mine does not make a great deal of water, more however in winter than in summer, and is handled by a bucket in one shaft, and by Cornish pumps in the other, 3 with 10-inch cylinders, and 1 with 6-inch cylinder; there is a 9-inch column for the 10-inch pumps, and a 6-inch column to the 6-inch pump. Giant powder, Nos. 1, 2, and 3, is used in the mine, principally No. 2; 1700 lb. are consumed monthly. The Patton, or main shaft, started in slate near the surface, passed into diorite, and continued in this formation to the bottom. The south shaft was sunk entirely in slate. Both shafts are timbered throughout. Sawed lumber costs at the mine \$22 $\frac{1}{2}$ (4*l.* 13*s.* 9*d.*) per thousand; and 5 ft. and 6 ft. lagging \$80-\$90 (16*l.* 13*s.* 4*d.*-18*l.* 15*s.*) per thousand.

The mine is about 20 miles from timber. The mine and the mill are in close proximity. The character of the ore is quartz, soft and easily crushed, free milling, with about $1\frac{1}{2}$ per cent. of sulphurets. The ore is crushed, pulverised under stamps, amalgamated in the battery and on outside plates, and the sulphurets are saved in Hendy pan concentrators, in riffle sluices, and on canvas. When running on rich ore, the coarse sulphurets which are caught in the sluices assay \$135 (28*l.* 2*s.* 6*d.*) per ton. From the same ore, concentrations from a Hendy assay \$110 (20*l.* 18*s.* 4*d.*) per ton. Fine sulphurets, caught on blankets, assay \$44 (9*l.* 3*s.* 4*d.*) per ton. In the first 5 months in 1888, before striking the rich ore, on the lower level—

	\$	c.	£	s.	d.
The sulphurets caught in Hendy pan concentrators assayed, per ton	96	33	=	20	1 5
The sulphurets caught in riffle sluices assayed, per ton	71	52	=	15	4 6
The sulphurets caught on canvas assayed, per ton	44	00	=	9	3 4
The upper tailings as they leave the mill assay, per ton ..	0	70	=	0	2 11
The lower tailings below the sluices assay, per ton	0	50	=	0	2 1

The 40-stamp mill is arranged for either water or steam power. A Knight's 6-foot water-wheel is used, under 254 ft. of pressure. A steam engine, with a cylinder 18 in. diameter, and 4 ft. stroke, connected with 2 54-inch boilers, 16 ft. long, is called into operation in case of failure of water supply. The boilers are supplied with a Freedman's injector, and Knowles' steam pump, No. 5. At the hoisting works there is a Blake's crusher, 9 by 15 in., and the ore goes broken to the mill. For one year with No. 6 and 7 straight slot screens, 3000 tons of ore per month were crushed. The order of drop in the Keystone mill is 1, 2, 3, 5, 4,—1 and 2 having $\frac{1}{2}$ in. more drop than the other stamps. In the next battery the order is reversed, so on throughout the mill. The inside copper plates are changed from one battery to another, to ensure a more even coating of amalgam on the plates with this drop. Shoes and dies are iron, and cost 4 cents (2*d.*) per lb. Weight of stamp: Stem, 310 lb.; tappet, 85 lb.; bosses, 215 lb.; shoes, 128 lb.; keys, 12 lb.; total, 750 lb. Wear of shoes and dies per ton: 40 shoes, 5120 lb.; 40 dies, 4360 lb.; total, 9480 lb., will crush 3000 tons of Keystone ore; 9480 lb. at 4 cents (2*d.*) = \$379·20 (79*l.* 0*s.* 10*d.*) less one-fourth weight returned, 2370 lb. at 2 cents (1*d.*) = \$47·40 (9*l.* 17*s.* 6*d.*); 3000 tons ore used up \$331·80 (69*l.* 2*s.* 6*d.*) in shoes and dies, or at the rate of 1·16 cent per ton. Running with a low feed, the Keystone drop wears iron faster than it is worn with other drops, but it is believed also to crush ore faster. The company tried chrome steel; but a set of shoes and dies lasting only 60 days, they were found not to be as economical as iron, where 2 cents per lb. are returned for old iron. The screens used are straight slot, No. 8, slightly inclined from the perpendicular. The plates on the aprons are 48 in. by 13 in. The width of the sluices, of which there are 2 to each 5 stamps, is 14 in. and the length 9 ft. The plate inside the battery is 48 in. in length. An end section of the block to which this plate is fastened varies from nothing to 2 in. on the perpendicular side, on the upper side from 50° to 85° from the horizontal. The plates used are copper, and have on the apron an inclination of $\frac{3}{4}$ in. to 1 ft., and in sluices an inclination of $1\frac{1}{2}$ in. to 1 ft. The percentage of free gold saved in the battery is about 75 per cent., and on the outside plates 25 per cent.

Loss of mercury per ton of ore worked: One year 1276 Troy oz. were lost monthly in mill, and 10 oz. in retorting = 1286 oz.; one year 861 Troy oz. were lost monthly in mill, and 10 oz. in retorting = 871 oz.; or an average of 1078·5 Troy oz. per month for two years. Allowing 14·58 Troy oz. for 1 lb. avoirdupois = 73·97 lb. avoirdupois monthly, at 70 cents (2*s.* 11*d.*) amounts to \$51·78 (10*l.* 15*s.* 9*d.*). Assuming 3000 tons of ore to have been worked, the mercury-cost per ton of ore worked would be 17 $\frac{1}{2}$ cent. The percentage of sulphurets is $1\frac{1}{2}$,

principally iron pyrites, with a little arsenical and antimonial sulphurets; they contain \$80-\$175 (16*l.* 13*s.* 4*d.*-36*l.* 9*s.* 2*d.*) per ton. Before striking rich ore last summer, the sulphurets for the first part of the year were running \$90-\$95 (18*l.*-19*l.*) for the fine and \$80-\$90 (16*l.*-18*l.*) for the coarse, the alloy of gold in the sulphurets containing about 83 per cent. gold and 17 per cent. silver by weight. They are saved in Hendy pans, sluices, and on canvas, and sold to the Amador Reduction Works. The charge for treating per ton is \$20 (4*l.* 3*s.* 4*d.*) for hauling, \$1 (4*s.* 2*d.*); the percentage of assay value received is 92; 1500 cords of wood are used per year, at \$6 (25*s.*) per cord for pine or oak. Both hoisting works are run by steam power. The mill requires 125 miners' inches of water; in all 180 inches are used at the mine and mill, at a cost of \$800 (160*l.*) per month. Milling costs per ton: Shoes and dies, 12 cents (6*d.*); labour, 27 cents (13½*d.*); water, 19 cents (9½*d.*); screens, 1 cent (½*d.*); mercury, 2 cents (1*d.*); other incidentals, 14 cents (7*d.*); total, 75 cents (3*s.* 1½*d.*)

Lockwood Mine.—This property is situated in the West Point Mining District, California. The course of the vein is north-west and south-east; the dip is westerly at the surface, but at a very small angle from the perpendicular, and at the lowest workings is vertical. The average width of ore throughout is about 3 ft. The mine is opened by 3 shafts, practically vertical; the north 350 ft. deep, the middle 165, and the south 135 ft. in depth. The hanging and foot-walls are granite. The vein is cut by a dyke of greenstone, 2-6 ft. thick, nearly at right angles to it, and pitching slightly to the south. This dyke has made no difference apparently in the character of the ore, which is the same on each side. The water, which is in no great quantity, is removed by a Hooker pump with 3½ inch column. Work is performed in the mine with single and double hand drills, and in the levels 1-2 ft. are run per diem; and in the shaft, which is a double compartment, 12½ by 4½ ft. in the clear, about 4½ ft. were sunk per week, at a cost for labour by contract of \$25-30 (5-6*l.*) per foot. Vein matter, quartz, &c., were passed through in sinking, and owing to the hard nature of the walls, not very much timbering has been required; in fact, the ore can be removed to the shaft without incurring injury or danger. Round timbers, squared on one side, were employed when needed, costing 5 cents (2½*d.*) per running foot for all sizes from 6 in. to 24 in. in diameter at the smaller end. For procuring timber, the mine is admirably situated, being almost in the heart of a dense forest. The company has built some 7 miles of ditch, and 1½ miles of road connecting the mine and mill. At present the ore is hauled in wagons, at a cost of 50 cents (2*s.* 1*d.*) per ton. The ore, which is hard and contains a large percentage of sulphurets, is treated by two different processes. After it has been crushed in a Blake crusher, about 7½ tons of ore are pulverised and amalgamated daily in a wet crushing battery, and amalgamated on

electroplated aprons and sluices, and concentrated on Frue vanners; 7-8 tons are taken from the rock breaker and pulverised dry in a Dodge mill, and treated by pan amalgamation in 3 different kinds of pans. The result of experiments, according to the statement of the superintendent, is that "stamps pay the best in the end," and the pulveriser is to be removed and replaced with 5 additional stamps. The mill is arranged for both steam and water power. The boiler is 42 in. by 16 ft., and the engine has a 12-inch cylinder and 24-inch stroke. Water is applied to a 4½-foot Knight's wheel, under a pressure of 110 ft. The Blake crusher has an 8 by 12-inch opening. The milling machinery consists of a Challenge feeder, one 5-stamp battery of 850 lb. stamps, one Dodge pulveriser, 2 Wheeler, 1 Boss, and 1 Fulton pan, 2 settlers, 8 ft. diameter, and 2 Frue vanners. The stamps drop 5-6 in., at the rate of 85 times per minute, crushing 1½ tons per stamp in 24 hours. At present steel shoes and dies are being tried as an experiment, and cost 9½ cents (4¾d.) delivered at the mill. No regular record has been kept as to wear of shoes and dies, but so far, it is stated, steel seems to be an improvement on iron in the matter of cost. The total capacity of the mill, stamps, and pulveriser is about 15 tons in 24 hours. In the battery about 1 miners' inch of water is used. The screens are slot punched, ¼ in. straight No. 10, 8 in. by 48 in. inside the frame, which is slightly inclined. The height of discharge is 7 in. when the dies are new. The mortar is 9 in. deep, from the bottom to the lip, and 12 in. in width. The diameter of the shoe is 8¼ in. The aprons are 4 ft. 7 in. wide and 5 ft. long, and the sluice is 30 in. wide and 12 ft. long. The plate inside the battery is 6 in. wide and 48 in. long; silvered plates are used inside, as well as on the aprons and sluices—the aprons having an inclination of 1¼ in. to the foot, and the sluices 1½ in. In the pans some potassium cyanide, bluestone, common salt, and concentrated lye are used, but there is no record kept of the quantity consumed, which however does not make a large item of cost.

Of the ore worked in the stamp mill, the recovery of free gold in the battery amounts to 75 per cent., and on the outside plates 25 per cent. The pans are run independently on the ore pulverised dry in the Dodge mill, and by this method about 90 per cent. of the value of the ore is saved, and the same amount by wet crushing, amalgamating, and use of Frue vanners, the tailings by either process assaying about \$2.26 (9s. 5d.) per ton. The ore contains about 6 per cent. of sulphurets, consisting of iron and copper pyrites, and arsenical, antimonial, lead, and zinc sulphurets, and is a general mixture, including some magnetic oxide of iron; their value is about \$50 (10d.) per ton. They were formerly treated at the company's works, in a muffle furnace 50 feet long, containing 8 muffles 8 ft. long, which had a capacity for about 4 tons a day,

and consumed 6 cords of wood, but since the erection of custom chlorination works near, the sulphurets are treated there, at a charge of \$20 (4*l.* 3*s.* 4*d.*) per ton, with an allowance of 90 per cent. of the gold assay. There are in the mine 24 miners, and 8 men in the mill, besides 1 foreman, 3 blacksmiths, 2 brakemen, 2 carmen, 1 carpenter, 1 machinist, 2 teamsters, 2 ditchmen, and 2 labourers, or 48 men in all. In the mine \$2.50 to \$3 (10*s.* 5*d.*–12*s.* 6*d.*) are paid daily, the same in the mill; labourers receive \$40 (8*l.*) per month, and board; carmen, \$2.50 (10*s.* 5*d.*) per day; brakemen, \$2.75 (11*s.* 5*d.*); carpenters, blacksmiths, and machinists, \$3–4 (12*s.* 6*d.*–16*s.* 8*d.*). For steam purposes at the hoisting works, 1½–2 cords of yellow pine and oak wood, costing \$2.50 (10*s.* 5*d.*) per cord are consumed; 2 cords are used per week in drying the ore for the pulveriser.

Loyal Lead.—This mine is situated in Drytown Mining District, California. The course of the vein is north, 30° west; the direction of the dip north-east, at an angle varying from 40° to 60°; the width of the main ore body is 16–45 ft. There are 3 tunnels on the mine, of the following respective lengths—118 ft., 300 ft., and 600 ft.—and reaching a vertical depth from the surface of 50, 70, and 120 ft. Both walls are diorite. The quantity of water coming into the mine is slight, making its appearance only in the bottom of a sump, 10 ft. below the level of the main ore chamber. There is very little timbering, as the ground does not require it at present. The timber used is round pine, and costs \$1.50 (6*s.* 3*d.*) per log, 16 feet in length by about 12 in. diameter at small end. The distance of the mine from timber is about 20 miles. The ore is transported to the mill by means of a tramway at a cost, including mining, of \$2.50 (10*s.* 5*d.*) per ton. Of this amount, 20 cents (10*d.*) is chargeable to transportation. The ore is crushed by stamps, and amalgamated in the battery and on outside plates. The sulphurets are caught in a riffle sluice and further concentrated in a Cornish buddle. The mill consists of 10 stamps, weighing 700 lb. each, with copper plates outside and in the battery. The power is water, under 260 ft. of pressure applied to a Knight's 8-foot wheel. One man is employed 6 days a week at \$3 (12*s.* 6*d.*) a day to run the car, or to deliver 98 tons of ore. The labour and incidentals bring up the cost of transportation to 20 cents (10*d.*) per ton. Cost of shoes and dies per lb., 4 cents (2*d.*); about one-fourth the original weight is returned at 2 cents (1*d.*) per lb. Wear of shoes and dies per ton of ore crushed, 14 cents (7*d.*). The last set of shoes and dies, weighing 1980 lb., was worn out in crushing 490 tons of ore, or about 4 lb. of iron were used in crushing 1 ton of ore.

First cost of 1980 lb. of iron, at 4 cents	\$	c.	79	20	=	£	s.	d.	16	10	0
Received for 500 lb. old iron, at 2 cents			10	00	=				2	1	8
Absolute cost of shoes and dies on 490 tons ore ..			69	20	=				14	8	4

It may be here stated that the ore is somewhat harder than the average ore on the mother lode. The quantity of water used in the battery is about 2 inches, and the mortar is 11 inches wide. Angle slot punched screens are used; slot $\frac{1}{2}$ in., No. 6. Starting with a 7-in. discharge above the new die, the screens are placed vertically, and have dimensions inside the frame of 42 in. by $5\frac{1}{2}$ in., and cost \$2 (8s. 4d.) each. One screen lasts during the crushing of 210 tons of ore. The cost of screens will be seen to be a little less than 1 cent ($\frac{1}{2}$ d.) per ton of ore crushed. Size of apron, 48 in. by 48 in.; width of sluice, 14 in.; length of sluice to each battery, 12 ft.; size of plates inside battery, $4\frac{1}{2}$ in. by 42 in. The plates are not silvered. Inclination, 2 in. to the foot. Feeding in the mill is done by hand. Loss of mercury per ton of ore worked, is $\frac{47}{100}$ oz. on the average. Concentration is done in 144 ft. of sluices.

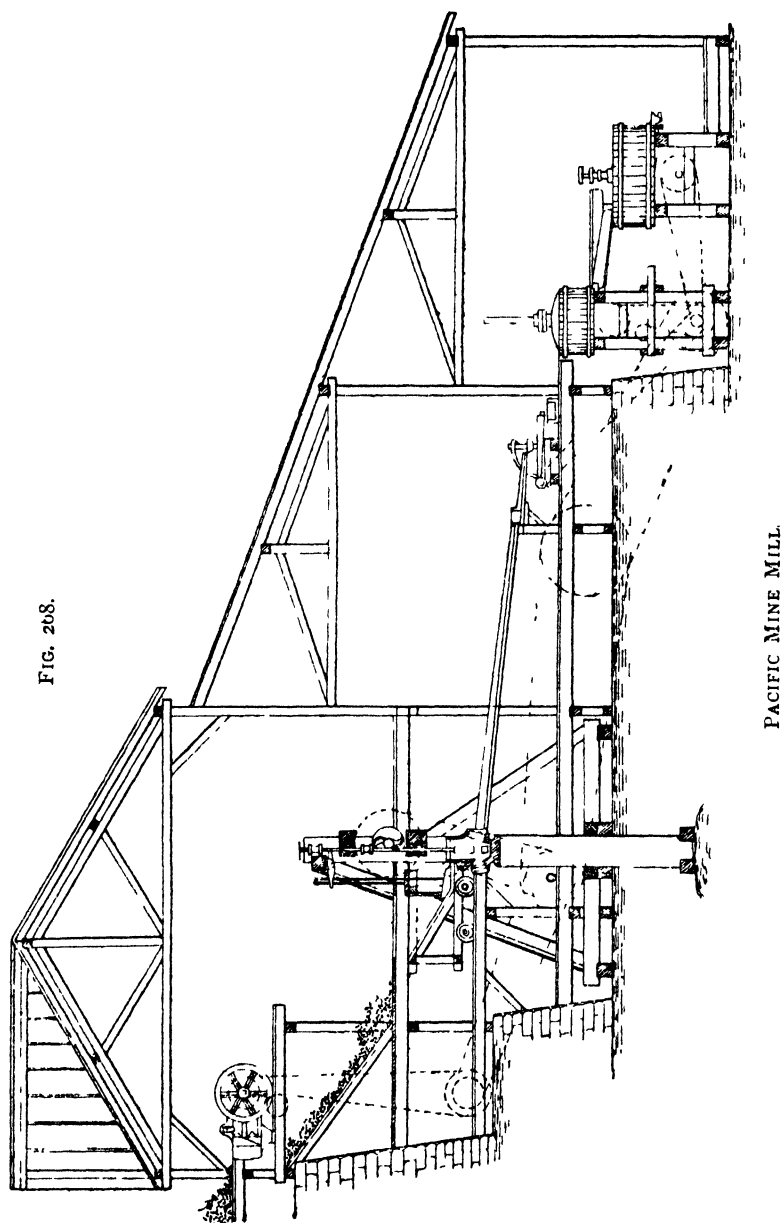
The sulphurets thus obtained are reconcentrated in a buddle. They consist of iron pyrites, with trace of galena, zinc blende, arsenic, antimony, &c. Sulphurets on the surface, near the mother lode, have not the value they contain at greater depths, owing to oxidation and freeing of a portion of gold contents, which appear as free gold. The sulphurets are sold to the chlorination works at Drytown, where \$20 (4l. 3s. 4d.) per ton is charged for the reduction, and the cost of hauling is 75 cents (3s. $1\frac{1}{2}$ d.) additional. The works return 90 per cent. of the assay value. The mine and mill are worked by the owners, who are miners and mill men. They allow themselves \$3 (12s. 6d.) each per day, and divide the surplus, on a co-operative plan, in proportion to the interest of each. An 8-foot Knight wheel drives the mill, for which 28 miners' inches of water are required, costing 20 cents (10d.) per inch. The Amador miners' inch of water is supposed to be 12 gal. per minute. It is measured with 4-inch pressure, above the top of a 2-inch slot in a 2-inch plank, giving about 12 lb. per sq. inch for friction, in 260 ft. The total cost of mining and transport is \$2 $\frac{1}{2}$ (10s. 5d.), and of milling \$1.45 (6s. 0 $\frac{1}{2}$ d.) per ton.

North British Mine, Maldon, Australia.—The quartz is crushed as it is taken out of the mine, and is discharged from the brace directly into hoppers. The stamper-boxes are hand-fed, and no mercury is put into them. The gratings through which the crushed quartz must pass have 200 holes to the sq. inch, and after passing through them the sand and water flows over two iron ripples, charged with 50 lb. of mercury. From these it flows over a copper plate, then over flat mercury wells; then over another copper plate, and then another flat ripple, and then another copper plate; thence it goes into a wooden shoot, which distributes the sand and water over blanket-tables, 14 ft. long. From these it is carried by a shoot into the tailings pit, and is then elevated by a sand pump, about 14 ft., and discharged into an iron tank, and from there is fed

over 3 large rocking cradles, each with 4 floors, 3 being lined with canvas, and the bottom one with baize. The tailings after leaving the cradles are conveyed by a shoot into a box which acts as a distributor over 3 tiers of strakes, 123 ft. long, and each $12\frac{1}{2}$ in. wide. These are skimmed every 4 hours. The sand and minerals saved from the blankets, cradles, and strakes are put on a platform, and from there are elevated by an air winch to the level of a reverberatory furnace, where they are all burned, after which they are run down an inclined plane to 2 Chilian mills of an improved description (the invention of Oswald) combining Chilian mill and arrastra in one. They are charged with 40 lb. mercury each. The discharge from the mills goes into a pit, and is kept for future experiment.

Pacific Mine.—This mine was formerly known as the Placerville, and is one of the oldest in California. The ore is the rock of the mother lode. It is stamped and amalgamated in the usual method of treating free gold ore. The tailings are concentrated with Hendy pan concentrators and Cornish buddles, blankets, and riffle sluices. At times the concentrations recovered are treated in a barrel, at other times by pan process, and sometimes sold to chlorination works. The mill is a water-power mill, containing a Blake crusher, next to the largest size manufactured, driven by a Morey 6-foot hurdy-gurdy wheel, under about 270 ft. pressure. The main machinery is driven by a Morey 8-foot hurdy-gurdy wheel, under about the same pressure. The mill grade was designed for thirty stamps, but 20 only were put in the mill. The weight of the stamps is 800 lb., drop 8 in., 82 drops per minute, and $1\frac{1}{2}$ tons were crushed in 24 hours. White iron, costing 5 cents ($2\frac{1}{2}d.$) per lb., was used for shoes and dies, and lasted, on the average, about 40 days, indicating a wear of about $4\frac{1}{2}$ lb. per ton of ore crushed, equivalent to about 21 cents ($10\frac{1}{2}d.$) per ton of ore; 4 inches of water are used in the battery. The screens, arranged vertically, are No. 6 round punched, 8 by 48 in. inside the frame. The plates are copper, and on the apron are about 30 in. by 48, and on the sluice 1 ft. in width by 16 ft. in length, having an inclination of $1\frac{1}{2}$ in. to 1 foot. In the batteries a front plate only is used, which is 6 in. in width, measured on the curve, and about 48 in. in length. The mill is furnished with Challenge feeders, and one combination pan and settler. In working the concentrations, 4 lb. sulphuric acid and 40 lb. salt are used in the pan. The usual amount of recovery in the battery is 75 per cent., on the outside plates 10 per cent., and in pans 15 per cent. The loss of quicksilver is about 2 cents ($1d.$) per ton of ore. The sulphurets, amounting to $\frac{1}{2}$ per cent, are arsenical pyrites of iron, containing 3 per cent. nickel, and assaying \$85 (17*l.*) per ton in gold and \$2 (8*s.* 4*d.*) per ton in silver. The ore mills \$6–\$18 (25*s.*–75*s.*) per ton, and the tailings average from a trace in gold to 75

cents (3s. 1½d.) per ton, no selection being made of the ore. At the hoisting works a friction gear, with 6-foot reel with flat rope, is run



either by steam power or water. When running by steam, a 30-ft. engine is employed ; when running by water power, the water power is

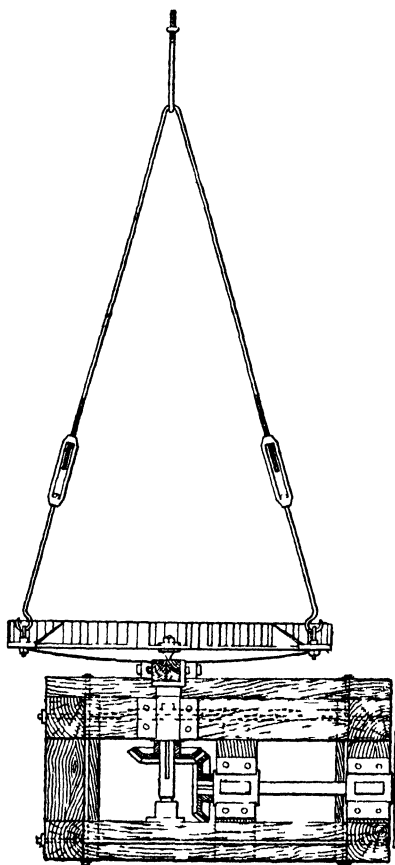
applied to a 4-ft. Knight wheel, under about 240 ft. of pressure. The price paid for water is 15 cents ($7\frac{1}{2}d.$) per miners' inch for 24 hours. Steam is only employed in case of accident to the water ditch. A section of the mill is shown in Fig. 262.

Plumas Eureka.—This mine, the property of an English company, and under the same general supervision as the Sierra Buttes, is situated in Quartz Mining District, California; the course of the vein is north and south; it dips to the west at an angle of 75° ; average width about 6 ft. The cost of mining and delivery of ore at mill is \$3.20 ($13s. 4d.$) per ton; cost of running tunnel, including timber, \$4.60 ($16s. 8d.-25s.$) per foot; average distance made, about 20 ft. per week. Pine timber is used, at a cost of 7 cents ($3\frac{1}{2}d.$) per running foot. Length of ditch built by the company, 8 miles. Means of transporting ore to mill consist of 3 tramways, two 1500 ft. in length, and one 800 ft. in length. Hanging walls, metamorphic slate; foot-wall, syenite. The ore is free milling, and contains about $1\frac{1}{4}$ per cent of sulphurets, valued at \$30 ($6l. 5s.$) per ton. These are treated by the chlorination process, at the company's own works, at a cost of \$10 ($2l. 1s. 8d.$) per ton.

This mine is equipped with a 60-stamp wet crushing mill, located on the west bank of Jamison Creek, at the foot of the mountain; it is provided with 3 Blake rock breakers, 12 Hendy feeders, 2 Patton pans, an iron barrel and automatic batea (as shown in Fig. 269) for cleaning up, 20 Hendy concentrators, 3 Duncan concentrators, 5 Patton concentrators, plated sluices, and a special set of plates in a separate "plate house."

The cost of milling is $48\frac{3}{4}$ cents ($2s. 0\frac{1}{2}d.$) per ton of ore. The mill employs 12 men, at an average of \$3.03 ($12s. 8d.$) per day. The ore yielded during 6 months ending June 30, 1888, an average of \$7.92 $\frac{1}{4}$

FIG. 269.



AUTOMATIC BATEA.

(1*l.* 13*s.*) per ton, this being above the general average for some years past. The mill tailings assayed \$2 (8*s.* 4*d.*) per ton for the six months. The loss of mercury per ton of ore is 0·27 oz. nearly. The gold varies in fineness; that from the batteries and first plates is about 700 fine, while that obtained by amalgamation of concentrates in pans, and from the lower plates, varies from 500 down to ·018 in fineness. In the latter case it contains lead, extracted, no doubt, from lead sulphate and carbonate in the concentrates. The mill is driven by water, brought through 4 miles of flume from Jamison Creek, and under a pressure of 260 ft., to a Knight wheel of 8 ft. diameter, furnished with an adjustable nozzle, by which variations in the quantity of water obtainable may be accommodated. There is also a 75 H.P. steam engine, used when required to supplement or replace water power. The data given as to cost of working apply to the use of water power, that of steam being considerably more expensive. The stamps weigh 850 lb. each, drop 8½ in. 80 times per minute, and crush 2·53 tons of ore per stamp in 24 hours. The shoes and dies are of steel, mostly English, and cost 8 cents (4*d.*) per lb. The wear of shoes and dies is 1·06 lb. of steel per ton of ore crushed, and the cost, therefore, is 8½ cents (4¼*d.*) nearly. The screens are diagonal slot punched, No. 8, with a discharging surface to each battery of 45 in. in length and 6 in. high. The foot-rail of the screen frame is 6 8 in. high, the former height being used with dies that are half worn, the latter with new dies, the object being to keep the discharge at as nearly the same height above the dies as possible. Each mortar is furnished with a lip plate and a cast-iron trough, which receives the pulp and conducts it to a plated sluice 15 in. wide and 20 ft. long, in sections of 15 and 5 ft. respectively, with a drop of several inches from the one to the other. All these plates are silvered. Inside front plates are 5 in. wide.

The percentage of recovery of free gold in the batteries is 60, the remainder being found on the plates. The amalgam yields 20 per cent. of its weight in gold when retorted. After passing the plates, the pulp is distributed among the concentrators. The product of 16 of the Hendy machines is reconcentrated by 4 more of the same, that of 2 Duncans by a third Duncan, and 4 of the Pattons in like manner supply a fifth. The waste from the reconcentration is again passed over silvered copper plates. The whole of the tailings, after leaving the concentrators, flow to the plate house, where the stream is divided between 36 plated sluices, each 15 in. wide and 5 ft. long, being diluted by an addition of clear water on each plate. The plate house yields about \$300 (60*l.*) worth of amalgam per month, rather more than paying the expense of keeping the plates in good condition by frequent re-silvering, which is done economically in the company's works. After

all this, the tailings still contain a value of \$2 (8s. 4d.) per ton, and furnish profitable employment to about 30 arrastras. As the arrastra men subject the material to a slight concentration in broad ripple sluices, allowing the slimes to flow off before regrinding the coarser material, it seems probable that the value is mostly in the form of free gold enclosed in the particles of quartz, and consequently could not be recovered in the mill without finer crushing, which, however, would be disadvantageous in other ways. The sulphurets are treated by amalgamation in the Patton pans, yielding \$25.39 (5l. 5s. 9d.) per ton; the cost of treatment was not given.

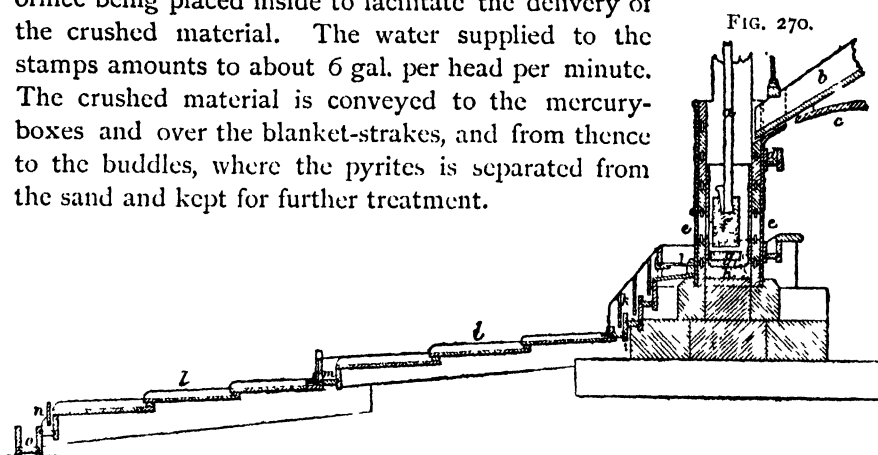
Port Phillip and Colonial Co.'s mill.—The Port Phillip and Colonial Gold Mining Co.'s works at Clunes, Victoria, Australia, are among the most perfect in the world, and their success is mainly due to the energy of the talented manager, R. H. Bland, who has very kindly furnished the following exhaustive account of the operations conducted under his direction.

"Operations at this mine were commenced early in 1857. The first battery of stamps was completed and got to work in May of that year, and comprised 20 heads of light stamps. These were rebuilt and added to from time to time as the supply of quartz increased. The present reduction plant comprises 80 heads of heavier stamps, 2 stone-breaking machines, and 7 buddles, driven by one 24-in. cylinder engine, supplied with steam by 5 multitubular boilers with a pressure of 60 lb., working expansive, and condensing, with an indicated horse-power of 127.

"In addition to the above there are 5 cast-iron barrels for amalgamating the blanket sand, and 2 steam-barrels for saving ground-up mercury, driven by a separate 12-in. engine; also a set of double-acting 10-in. pumps capable of lifting 600 gal. of water per minute, driven by a 10-in. engine.

"Stamping batteries.—These are built on the old principle, fitted with square-shaped stamps; 24 of these weigh with the lifters about 8 cwt. each, the remaining 56 weigh about 6½ cwt each. The lifters are made of 2½ in. bar-iron, are driven at a speed of about 80 blows per minute, and crush an average of a little over 3 tons of quartz per head per diem of 24 hours. Fig. 270 shows the construction of the batteries: *a*, lifter; *b*, self-feeding hopper; *c*, spring; *d*, water-trough; *e*, escape; *f*, stamp-head; *g*, die; *h*, bottom box; *i*, loose bed; *j*, perforated plate; *k*, mercury-boxes; *l*, blanket-strakes; *m*, distributing-trough; *n*, mercury-box; *o*, waste-trough. The 'coffer' is of cast-iron of sufficient length for 4 stamp-heads. A layer of broken quartz is first placed in the coffer, and a wrought-iron block the width of the stamp-head and long enough for two heads is placed upon this, upon which the stamps work. By this method the cast-iron coffer remains uninjured, and it

has the further advantage of allowing space for the coarser gold to accumulate directly it is liberated from the quartz without being further broken up by the stamps. The escape plates, placed back and front, are made of stout copper, perforated with a tapered punch, the smaller orifice being placed inside to facilitate the delivery of the crushed material. The water supplied to the stamps amounts to about 6 gal. per head per minute. The crushed material is conveyed to the mercury-boxes and over the blanket-strakes, and from thence to the buddles, where the pyrites is separated from the sand and kept for further treatment.



PORT PHILLIP BATTERY.

"Pyrites.—The quantity of pyrites contained in the quartz from this mine is estimated at about $\frac{3}{4}$ per cent., equal to 15 cwt. to the 100 tons. After being concentrated in the buddles, it is sent to the furnace, and roasted sufficiently to get rid of the sulphur and arsenic. It is then taken to the Chilian mills and ground with mercury. This operation is carried on as follows. Each mill-pan is charged with about 1 cwt. of roasted pyrites and about 80 lb. of mercury, and sufficient water to damp the material; it is then ground for about 20 minutes. Water is then turned on and grinding continued until the ground-up pyrites is washed away, leaving the amalgam in the pan; the process is then continued. The material washed from the pan is passed through a concentrator, and thence over a cradle covered with blanket to further arrest any particles of ground-up mercury and amalgam that may have escaped the mills. The material so saved is placed in one of the steam-barrels and kept revolving for 2–3 days. After making a good many trials, we found the method we now adopt to answer the best. The bye-products, such as sulphur and arsenic, are, however, all wasted; but this cannot be avoided.

"Minc.—There are 2 engine-shafts at this mine. The north shaft is down 1193 $\frac{1}{2}$ ft., and is supplied with a winding-engine of 24-in. cylinder, and a very complete set of winding-gear, one 24-in. pumping-engine, and one capstan-engine, which also drives one of Root's blowers for driving air down the mine. These engines are supplied with steam from 4

tubular boilers. The south shaft is down 850 ft., and will be sunk at once to 900 ft. This shaft is supplied with a winding-engine and one boiler. The quartz from these two shafts is conveyed to the stamp-house by a very complete system of tramways. Four quartz lodes appeared on the surface at this mine, called respectively the 'Western,' 'Robinson's,' 'Old Man,' and the 'Eastern.' Subsequently another lode was discovered under the basalt and called the 'Welcome,' which, however, did not continue payable much below No. 5 level. Another vein called the 'New Eastern' was passed through in sinking the north shaft near No. 9 level, and, though small, yielded well for two levels. The Old Man and Western lodes have continued the most defined in depth, and though very variable, continue to yield fairly. This mine was worked almost wholly on the contract system until 1874, when the tribute system was commenced, and has been continued to the present time, the results being more satisfactory to all parties interested in the mine. The miners have the opportunity of exercising skill and industry, and are paid in proportion to the results.

"(Signed) R. II. BLAND."

Rio Grande mill.—Fig. 264 shows a gold-mill plant for mule-power, designed for the Rio Grande do Sul Gold Mining Co., Brazil. The mill cost 730*l.*, and crushes 7 to 8 tons of gold-quartz per 24 hours. The weight of stamp-head is 160 lb.; diameter, 5½ in.; depth of shoe, 4½ in.; throw of crank, 8 in. The dimensions of the gratings are two 16¼ in. by 13½ in., and one 16¼ by 11¾; the perforations number 144 per sq. in., and their diameter is 0.01 in. The number of blows per minute is 150; fall of head, 12 in.; weight of heaviest piece, 7 cwt.; framework, wood; water required, 2 cub. ft. per minute; power required, 8 mules.

St. Mungo Company.—This plant, erected by Middleton and Delbridge, engineers, of Sandhurst, Australia, consists of a battery of 30 revolving stampers, 5 heads in each box. The whole of the machinery, including the water and tailings pumps, is driven by a horizontal steam engine, having a cylinder 22 in. diameter, with a stroke of 4 ft. 6 in. The engine is fitted up with variable expansion gear, and Tangye's patent governor. A double-acting horizontal air-pump is used, the piston-rod being coupled direct on to the outer end of piston-rod of the steam engine. The engine is erected so that it is capable of driving, when required, an additional battery of 30 heads. Steam is supplied from 2 boilers, 28 ft. long, having each a diameter of 6 ft., constructed with Galloway's patent tubes and water bridge in the firing tubes. The boilers are fed with water by one of Tangye's patent steam pumps, which is a very great improvement on the old method of feeding by a pump attached to the steam engine; by using the steam pump, the

to the engine is $7\frac{1}{4}$ in. diameter, the lengths of shafting being secured together with strong cast-iron turned flange couplings. At front of frames or horses are fixed cast-iron brackets for carrying the water pipes, and a very substantial footboard or platform; a strong hand-railing of wrought-iron gas piping is fixed to the outer edge of the footboard. The stamper boxes are of the latest improved design, being fitted with double splash doors of wrought iron, wood covers, and inside splash boards. On tops of covers are fixed gun-metal pipes, &c., for conveying feed water into boxes. The stamper boxes are provided with false bottoms of the best scrap hammered iron. The shanks are $3\frac{1}{4}$ in. diameter, screwed to receive discs $9\frac{1}{2}$ in. diameter. The cams are strong wrought iron, Seymour's patent. The ripple and blanket tables are very substantial, fitted with cast-iron wells for mercury, the principal timber used in the tables being Californian red pine. At the lower ends of blanket tables are fixed Halley's patent percussion table; after the tailings have left these tables, they are raised by a tailings pump, and conducted by shutes to be further treated by cradling, &c.

The water for supplying the battery and injection for condenser is raised by a plunger pump; the water is raised into a large wrought-iron tank placed about 3 ft. above the centre of the engine. Both tailings and water pump are worked by separate bell cranks and sweep rod; this allows the pumps to work at any length of stroke found necessary. The steam engine is erected upon a substantial bed of brick and stone work, the engine platform is elevated so high from the flooring of house that it enables the engineer in charge to overlook the whole of the machinery at work. The buildings over battery engine and boilers are enclosed and covered in with galvanised corrugated iron. The whole of the flooring of main building of battery is laid with asphalt.

Sierra Buttes Mine.—This mine, the property of an English company since 1870, and worked since 1850, is situated in Placer county, California. The vein runs generally east and west, dipping 41° north, and averaging 20 ft. wide. The water supply is drawn from several lakes, and is brought $7\frac{1}{2}$ miles in a flume to the mine. The walls in tunnel No. 9, at 7000 ft. from the entrance, are, on the hanging side, metamorphic talco-siliceous slate, and on the foot-wall metamorphic calcareous slate. The entire length of tunnel, except about 800 ft., is timbered with fir, costing 9 cents ($4\frac{1}{2}d.$) per running foot for round logs of 18 in. average diameter.

The ore is extracted at a cost of \$3.54 (14s. 9d.) per ton, including dead work, and is treated by wet stamping, amalgamation, and concentration. It consists of quartz with free gold, and about 9.6 per cent. of sulphurets, consisting of pyrites with a little galena and blende, the concentrations assaying about \$97 (20l. 4s. 2d.) per ton in gold, and \$3

(12s. 6d.) per ton in silver; the yield of free gold from the rock^{matter} given. The concentrates are treated by chlorination at the company's works at a cost of \$16 (3l. 6s. 8d.) per ton, including the expense of concentration, which, considering the low percentage of sulphurets in the ore, is probably in the neighbourhood of \$10 (2l. 1s. 8d.) per ton of concentrates, leaving about \$6 (25s.) per ton for the actual expense of the extraction of the gold, consequently the stated cost of milling, namely, 34½ cents (17¼d.) per ton, does not include the concentration. The proportion of the gold saved from the concentrates is 93 per cent. of the assay value. The assays of mill tailings indicate that about 76 per cent. of the value of the ore as it comes from the mine remains in the mill as amalgam and concentrates, which, considering that the ore now being worked is of such low grade that the arrastra men will not touch the tailings, may be regarded as a good result; for these men say that when the ore pays as much as \$5 (20s. 10d.) per ton in the mill, it will pay to take up the tailings and regrind them in water-arrastras. But although there are several arrastras in operation here, they are working on a lot of old tailings and not of recent production.

The principal mill has 60 stamps of 850 lb. each, dropping 8 in. 86 times per minute, and crushing 2¼ ton of ore to the stamp in 24 hours, through No. 7 slot-punched screens, having a discharge surface to each battery of 5 stamps, 48 in. in length by 8 in. in height, set vertically. The shoes are of steel, costing 7½ cents (3¾d.) per lb.; the dies of cast iron, at 4½ cents (2¼d.) per lb. A shoe weighs 150 lb. and lasts 100 days. A die weighs 100 lb. and lasts 45 days. The consumption of steel per ton of ore crushed is therefore $\frac{645}{1000}$ lb., costing 6·56 cents (3¼d.) per lb. The consumption of iron dies per ton is $\frac{926}{1000}$ lb., costing 4½ cents (2¼d.) per lb. It is not to be inferred that iron is cheaper than steel, because the shoe necessarily wears more than the die, which is partly protected by a layer of ore. The total cost for shoes and dies is 11·06 cents (5½d.) per ton of ore crushed, which, however, may be reduced in this case for the value of the residues, amounting to 20 per cent. of the weights, which bring 2 cents (1d.) per lb. as old iron. This leaves the actual expense for shoes equal 6·31 cents, and for dies, 4·13 cents; total, 10·44 cents (5¼d.) per ton of ore.

There are no aprons in the usual meaning of the term; instead, there is a lip-plate on the mortar, which is 4 ft. in its larger dimensions by 1 ft. in the shorter. The pulp, passing through the screen, flows on this plate and into a cast-iron trough about 1 ft. wide by 4 ft. long, extending across the front of the mortar, under the lip, and attached to the mortar by flanges and bolts. The bottom of the iron trough is inclined in the direction of its length, that is, towards one side of the mortar, at such an angle as to cause the pulp to flow that way and to pass through a

Mortar in front of the trough, into a distributing box and thence through holes into the plated sluice, which is 15 in. wide and 20 ft. long. Each battery of 5 stamps is thus arranged. The usual style of apron was tried and discarded, as it was found that so large a surface could not be kept in good condition. The present arrangement has the advantage of leaving the front of the mortar easily accessible.

• It is a singular fact, and one which calls for investigation, that silvered plates cannot be used in this mill, as the silver comes off so soon that the cost of such frequent replacing would be too great; while on the other hand, silvered plates are considered to be indispensable in the 20-stamp mill of the same company. The only difference known, as to the two cases, is, that this mill (the lower) is crushing ore from the deep workings of the mine, containing a small percentage of sulphurets, while the upper mill treats surface ores with no sulphurets. Inside plates are used in the mortars. The back plates are 3 in. high and as long as the mortars; the plated "chuck-blocks" in front are about 6 in. high, the foot rails of the screen frames being 8 in. high, and the screen surface the same. This leaves the top rail of the screen frame below the top of the opening in the front of the mortar by a space of about 3 in., through which a view can be had of the interior of the mortar at any time, a very good arrangement, because it sometimes happens in a mill, that a shoe becomes detached and remains vertically in place in the mortar, allowing the boss to slip over the shank at every stroke, and thus the stamp continues dropping to the full extent; and the only indication of the accident is the sound of metallic contact, which, in one stamp among sixty, might escape the notice of the attendant for some time, leading to the ruin of the boss, as well as wasting the time of the stamp. Of the recovered free gold, 80 per cent. is saved in the batteries, and 20 per cent. on the plates. An iron barrel, supplemented by an automatic "batea," also of iron, and carrying 3 cannon balls, is used for cleaning amalgam.

The quantity of water used in the batteries is not measured. The loss of mercury per ton of ore worked is not given, but is much greater in the lower than in the upper mill, probably from the same causes that led to the observed difference in the action of the plates. There are 24 Frue vanners in the mill, using in all 4 "inches" of water measured under 6 in. pressure. The company's other mill of 20 stamps on the mountain is employed in working old dumps and surface rock, at a cost of 75 cents (3s. 1½d.) per ton all told.

In the chlorination works there is nothing unusual to note, except that, owing to complaints of the effect of the fumes from the roasting ore on vegetation in the neighbourhood, steps were taken to abate this annoyance, as follows:—The smoke and fumes from the furnace, instead

of being allowed to escape at the top of the stack, are drawn through an iron flue, and afterwards forced through an earthen flue by means of a "water-blast," consisting of a jet from $\frac{1}{4}$ -inch nozzle under 530 ft. fall. After passing through 16 ft. of the earthen flue, the fumes, with the water from the jet, go down a vertical flue about 22 ft.; through this flue also flows a stream of about 10 in. of water in the form of a shower. The water and such portion of fume as is not yet absorbed, now traverse a nearly horizontal wooden conduit or pipe, about 120 ft., to another short fall, where still another shower of water falls with it, and this is succeeded by another horizontal flue. When the final exit is reached, only a slight smell of sulphur can be perceived in the escaping steam and gases.

The roasting furnace is a long reverberatory with several hearths. The total dimensions are 50 ft. long by 12 ft. wide inside. The capacity is 2 tons of concentrates roasted in 24 hours, with a consumption of $\frac{1}{2}$ cord of pine wood per ton of ore, at \$4.25 (17s. 8 $\frac{1}{2}$ d.) per cord.

The entire establishment is furnished with incandescent electric lights, for the production of which a Knight wheel of 8 ft. diameter is employed, while the batteries and Blake rock breakers, of which latter there are 3, are worked by means of a Pelton wheel of 6 ft. diameter under 530 ft. fall. The concentrators are operated by another Pelton wheel of 3 ft. diameter under the same pressure, and a 6 ft. Pelton, with 60 ft. of pressure, pumps the "hypo" in the chlorination works for the leaching of the small quantity of silver which the concentrates contain.

Zeile Mine.—This mine is situated in Jackson District, California, and was formerly known as the Coney. The walls are of slate, with gouge on the footwall, from a few feet to 55 ft. in width. During the summer months the mine yields 15,000 gal. of water daily, and in winter 15,000 to 50,000 gal. No pumps are used, but in their stead tanks of 400 gal. capacity, self-filling and self-discharging, running on wheels, are employed in keeping out the water. The ore is quartz and slate, with a small amount of free gold, and yields about 2 $\frac{1}{2}$ per cent. of very pure iron sulphurets. A large rock breaker at the hoisting works receives and crushes the ore as fast as it is raised from the mine, and the ore goes to the mill crushed fine enough for the stamps, without having had an attendant at the rock breaker. Amalgamation is done in the battery and on the outside plates. The sulphurets, obtained by concentration, are chlorinated by the Plattner process, in works belonging to the company, which have a capacity for treating 3 tons in 24 hours. The mill is arranged for both steam and water power. When steam is used, which is only in case of a failure of the water supply, 2 boilers, 54 in. diameter by 16 ft. long, and a 16-inch cylinder engine, Corliss pattern, are employed, and the power is transmitted to the main shaft by five

1½ in. manilla ropes, running over groove pulleys. When running by water, a 6-ft. Pelton wheel, with a rubber adjustable nozzle, under 145 ft. pressure, drives the mill, compressor, and blower, and the power is transmitted by a 5-ply rubber belt of suitable width. The rubber adjustable nozzle has been in use nearly two years, and works to entire satisfaction. It is made of perfectly smooth medium rubber, and shows less wear than iron nozzles which have been in use for the same length of time; the same nozzle, under the same pressure, can be used in running any number of stamps, from 5 to 40, by turning a screw, which compresses the rubber endwise and enlarges or diminishes the opening. Chrome steel is now used for shoes and dies, and the company pays per lb. according to wear. The quantity of water used in the battery and concentrators has never been measured, but is estimated at 20 miners' inches in 24 hours. Screens No. 4, straight ½-inch slot, made of No. 12 Russia iron, placed vertically, are used, with dimensions, inside frame, 6 in. by 50 in.

The size of the aprons is 58 in. by 30 in., the width of the sluices is 16 in.; the length is 12 ft.; the inclination of the apron is 2½ in.; of the sluice 1½ in. to one foot. One-third of the total recovery of the ore value is saved on the inside and outside plates as free gold, and two-thirds on the concentrators as sulphurets.

There are 16 Frue concentrators operating in the mill, which use, by estimate, 5 miners' inches of water daily. The gold caught in the mill is worth about \$17.75 (3*l.* 13*s.* 11*d.*) per oz.; that recovered in the chlorination works is generally 998 fine. The chlorination works are provided with every appliance for successfully working sulphurets by the method the name implies. The roasting furnace, with a capacity for treating 3 tons of sulphurets in 24 hours, is 7 ft. high, 11 ft. wide, and 66 ft. long, outside measurement, with a dust chamber 6 by 6 ft. square: a stack 5 by 5 ft. square at the base, and 3 by 6 ft. at the top, 40 ft. high. The furnace inside has 3 hearths, with 2 drops of 6 in. each. It is lined inside with fire-brick, and shaped so that the charge may be easily rabbled. The consumption of wood is 1⅓ cord a day in summer, and 1⅔ cord in winter. About 1 ton of sulphurets, mixed with 5 lb. of salt, is charged every 8 hours. A charge is drawn about every 8 hours, making about 3 tons worked per day. A specimen charge for gasing Zeile sulphurets is given as follows: For 4½ tons of sulphurets, 60 lb. manganese peroxide, 60 lb. salt, 100 lb. sulphuric acid at 66°.

After the gased pulp has been leached, a further addition of a few lb. of sulphuric acid is made to remove perceptible bases. The object of removing these bases is to give a cleaner or purer gold product. The base most commonly precipitated is lead, as sulphate; if present

and not precipitated by sulphuric acid, it would be precipitated when iron sulphate is added to precipitate the gold, thus contaminating it. Some lime sulphate is precipitated also, and other bases mechanically go down with lead sulphate in small quantities. By this treatment the gold product is purer, and less iron protosulphate is used. The liquid is run off into the precipitating tanks, and the gold is precipitated with iron sulphate in the usual manner.

There are employed in and about the premises, exclusive of the superintendent, 107 men in the following manner: 85 men and 2 foremen in the mine, 4 men and 1 foreman in the mill, 13 men on outside work, and 5 men in the chlorination works. The average wages paid are \$2.50 (10s. 5d.) a day for miners; \$2 (8s. 4d.) and \$2.25 (9s. 4½d.) for ore sorters and carmen; \$2.75-3 (11s. 5½d. to 12s. 6d.) for timber men; engineers, \$3.50 (14s. 7d.) and \$2.50, and for carpenters \$3.50. Four cords of wood per day are used for steam purposes, and 1½-1¾ cord for roasting sulphurets. First class wood costs \$5.50 (22s. 11d.) per cord; white oak, \$5 (20s. 10d.); black oak, \$6 (25s.); pine, \$3-5 (12s. 6d.-20s. 10d.) Big yellow pine split, such as is used in the roaster, costs \$5.50 per cord. Steam gearing at the hoisting works consists of 2 double engines, Corliss pattern, 12-inch cylinder, 2 boilers 54 in. by 16 ft. long, connected or single; reels 5 ft. diameter; round cable, 1½ in. flexible steel, which lasts about one year, is used. Water machinery at the hoisting works consists of a Pelton 4-ft. wheel under about 100 ft. pressure running the sawmill and framing machine. Water is purchased of the Blue Lakes Water Company on private contract.

CHAPTER X.

BIBLIOGRAPHY.

THE following form a large proportion of the books, periodicals, and maps consulted during the preparation of this volume and its predecessor. It will be found that many of the papers contributed to the proceedings of scientific societies or to technological newspapers are quoted under the authors' names. Special references to miscellaneous notes in the latter publications would be practically impossible: indeed, such papers as the *Mining Journal* and *Mining World* of London, the *Engineering and Mining Journal* and *Mining Record* of New York, the *Mining and Scientific Press* of San Francisco, the *Mining Review* of Chicago, and the *Mining Industry* of Denver, are indispensable to all having a direct interest in this subject.

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 WYLD (J.) South African Diamond- and Gold-fields.

CHAPTER XI.

GLOSSARY AND USEFUL TABLES.

- Almócfre* (U. S. Colombia), a kind of hoe used in placer mining (p. 151).
Altai, &c. (Mongol), gold.
Amalgamating barrel, see p. 526.
Amas (Malay), gold.
Amas lichin (Sumatra), nugget-gold.
Amas muda „ inferior gold.
Amas süngei-abu „ nugget-gold.
Amas supayang „ vein-gold.
Amas urci „ gold-dust.
Amurang (Ceylon), gold-ore.
Anna (India) = 1½ d.
Apitascudes (Latin), roasted and powdered quartz.
Arany (Hungarian), gold.
Areng (Borneo), pay-dirt, which is a yellowish gravelly earth, sometimes containing also diamonds.
Arrastra, a circular enclosure paved with stones, about 12 ft. diam., on which mineral is ground by mules dragging round large heavy stones. It is a very slow and crude apparatus. See p. 486.
Arroba (Brazil) = 32·37 lb.
Arrugia (Spain), deep gold mines.
Aruppukarans, a gold-washing caste in Madras.
Aur (Roumanian), gold.
Aurum (Latin), gold.
Aventadero or *aventador* (in Peru), a slide of loose ground containing alluvial gold.
Bahar (Malay Pen.) = 4 cwt.
Baluca (Latin), fine grains of gold.
Banket (S. Africa), auriferous alluvial conglomerate, allied to the “cement” of Australia, but in which the gold is said to have been precipitated from solution by water percolating the beds.
Bar, a hard ridge of rock crossing the bed of a stream, on the upper side of which gold is likely to be deposited.
Barranco (Venezuela), a mining shaft.
Barrel quartz, quartz rock having a corrugated form.
Batea, a gold-washing bowl (see p. 151).
Battery, a set of stamps (see p. 426).
Battu-uji (Malay), touchstone.
Batu kawi (Sumatra), a red stone supposed to be an infallible sign of gold.
Bed-rock, the solid hard rock underlying loose and incoherent strata.
Bench, a terrace on the side of a river, and having at one time formed its bank.
Auriferous benches are termed reef-wash (q. v.) in Australia.
Beneficio (Argentine Rep.), productive ore.
Beting, quartzose gold matrix of the Malays.

Bezer (Heb.), gold ore.

Bilong, the Malayan adze.

Blanket-table, see p. 516.

Block reefs (Australia), reefs showing frequent contractions longitudinally.

Bolivar (Venezuela) = 1 franc or $9\frac{1}{2}d$.

Bonanza, an aggregation of rich ore in a mine.

Bongkal (Straits Set.), a gold weight = 832.84 gr.; 20 *bongkals* = 1 *catty*.

Bottom, see Bed-rock.

Box (Amer., Aust.), a 12- or 14-ft. section of a sluice.

Brownstone (Australia), decomposed iron-pyrites.

Buck stone (Australia), non-auriferous rock.

Buddle, see p. 548.

Bungarroo (Telugu), gold.

Cacho de bateador (U. S. Colombia), a boat-shaped piece of horn used in gold-washing.

Cacó (Brazil), white sugary quartz.

Cajon or *Caxon* (Bolivia) = 50 *quintals*.

 " (Peru) = 60 "

 " (Chili) = 64 "

 1 *marco* of gold per *cajon* of ore = 2 oz. 14 dwt. 10 $\frac{3}{4}$ gr. per ton.

Canalicium (Latin), gold obtained from vein workings.

Candareen (China) = $\frac{1}{10}d$.

Canga (Brazil), a rock composed of sharp-cornered, angular (rarely slightly rounded) fragments of micaceous iron, specular iron, and magnetic oxide of iron, held together by red, yellow, or brown ochreous cement; it is often very auriferous, and sometimes contains scales of talc and chlorite, and stray fragments of *itacolumite*. It is widely and deeply distributed in valleys and on hill-slopes and summits, and has resulted from glacial action.

Canoa (Brazil), a platform used in gold-washing.

Carga (Mexico), variously given as 3 *quintals* = 312 lb., 12 *arrobas* = 300 lb., and 12 *panegas* of 12 *almudas* = 18 bushels.

Cascayo (Venezuela), a decomposed schist, on which the pay-dirt lies.

Cascalho (Brazil), gravel, which, when auriferous, is composed of quartz fragments.

Cascalho virgem, the deeper and older gravel of a river-bed.

Cash (China) = $\frac{1}{100}d$.

Casing, material found between a reef and its walls.

Catty (China) = $1\frac{1}{4}$ lb. av.

Catty (Straits Set.), a gold weight = 2.9818 lb. troy.

Cement, very widely applied to all auriferous conglomerates.

Cerro (Spanish), a rocky hill.

Chinna (Kanada), gold.

Chacra (Bolivia), an inheritance of gold.

Chaṅgkul (Sumatra), a miners' hammer.

Chocque (Aymara), "gold," whence the local names Chuqueapo (now La Paz), Chuqueaguillo, Chuquesaca (now Sucre).

Chonkole, the Malayan spade.

Clavo (Mexico), a very rich spot of small horizontal extent, but constant in depth, i.e., a "pay chimney."

Colour, minute traces or individual specks of gold.

Contour-race, a leat or water-course following the contour of the country.

Country or *country-rock*, the geological strata on each side of a reef or vein.

Cradle, see p. 156.

Creviceing, picking out the gold caught in cracks and crevices in the rocks over which it has been washed.

Creadero (Span. Amer.), an "indication" of the existence of gold.

Crusade = $\frac{1}{8}$ oz. of gold, value 2s. 6d., or = 400 *reis* = 1s. 9½d.

Curi (Ecuador), gold, whence Curaráy river.

Dalama (Zambesi), gold.

Daric (Persian) = 123·7 gr., value 1l. 1s. 10½d.

Deheb (Malta), gold.

Denounce (Mexico), to obtain a mining grant for a certain spot.

Dip, the downward inclination of a vein.

Doilia (Russia) = 0·685 gr.

Dolly (Australia), a springing pole about 50 ft. long, fixed to the tops of posts set in the ground, and having at the small end a rope, handle, and shoe attached, for smashing rock on an anvil put beneath. The spring of the pole greatly reduces the labour. Another form is used in the Philippines. See p. 118.

Dorongee [*durunzi*] (Assam), a gold-washing trough.

Drift, (1) a level; (2) very loose, friable, alluvial deposit, requiring close timbering to work; (3) glacial drift, alluvial deposits formed by ice.

Drifting, working a mine by means of driving tunnels on it.

Duin, gold-washing dish used in Jashpur, India.

Dulan (Borneo), circular concave trays for washing gold.

Dump, the fall available for disposal of refuse at the mouth of a mine or end of a gravel working.

Faiscadore (Brazil), a gold-washer.

False bottom, (1) a movable bottom or floor in a buddle, battery, or sluice; (2) in alluvial mining, a stratum on which pay-dirt lies, but which has other bottoms below it.

Feather-edge (New Zealand), a local term for the passage from a false to a true bottom.

Float-gold, gold in tiny thin scales, which float on running water.

Floating reef, masses of bed-rock found displaced and lying amongst alluvial detritus.

Florin (Holland) = 1s. 8d.

Floured mercury, mercury which has become granulated and each grain coated with a film of sulphide, destroying its amalgamating power (see p. 474).

Flour-gold, the finest alluvial gold, sometimes found as a coating on quartz pebbles in cement.

Fluming (America), lifting a river out of its bed into a wooden launder or flume, in order to get at the bed for working.

Fossicking, almost identical with creviceing (q. v.).

Fuang (Cochin China) = 3½d.

Galimador (Calif.), a kind of reverberatory furnace.

Gamella (Brazil), a wooden bowl, about 2 ft. wide at the mouth, and 5 or 6 in. deep, used for washing gold out of the auriferous matter collected in sluices and in river-sands.

Garimpeiro (Brazil), a gold-seeker and smuggler.

Gogo (Philippines), a plant whose juice is said to catch fine gold.

Goud (Dutch), gold.

Gouge (Nova Scotia), a narrow band of slate next the vein, which can be extracted by a thin, long, pointed stick, and yields gold.

Grade, the amount of fall or incline given to ditches, sluices, tunnels, strakes, &c.

Grating, a perforated iron sheet or wire webbing, enclosing the stamper-box.

Gravel, any broken-down rock, auriferous gravel being generally quartzose.

Greda (Venezuela), pay-dirt—a yellow ferruginous clay, containing nuggets and small grains of gold.

Ground-sluice, see p. 169.

Gua (Huarpe), gold; hence *guachi*, mountain of gold; *gualilan*, land of gold.

Guaca (U. S. Colombia), a mining tunnel.

Guld (Dan., Swe.), gold.

Gutter, the lowest portion of a lead, which contains the most highly auriferous dirt.

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Hade, dip.

Han (Siam), gold.

Hatter (New Zealand), a miner working on his own account.

Head of Water, the quantity consumed by any undertaking. See *Miners' inch*.

Headings, coarse gravel or drift overlying the wash-dirt.

Head-race, an aqueduct for bringing a supply of water.

Hemma (Sanskrit), gold.

Honna (Kanada), gold.

Ita, Japanese gold-washing board.

Itabirite (Brazil), a rock composed of micaceous specular iron-ore (rarely laminated) and a little oxide of iron and manganese with quartz disseminated; in the pulverulent form it is called *jacutinga*. Both carry free gold.

Itacolomite (Brazil), a rock composed chiefly of fine-grained quartz, united by thin laminae of chlorite and talc.

Itambamba (Brazil), a plant (? *Solanum sp.*) whose juice is thought to help catch fine gold.

Jacutinga (Brazil), a pulverulent variety of *itabirite*.

Jhoras, an Indian gold-washing caste.

Joren, a scoop-shaped bamboo basket used for carrying auriferous gravel in Japan.

Julgars, an Indian caste whose employment is gold-washing.

Kanchana (Malay), gold.

Katouti, gold-washing trough of the N.W. Provinces, India.

Kisye (Malay), rattan sieves used in gold-washing.

Ko-gane (Jap.), gold.

Kopeck (Russia) = 0.38d.

Kua, specially shaped hoes used for working gravel in the sluice in Japan.

Kura (Maori), gold.

Laminae (Span. Amer.), scale gold.

Lavadero (Span. Amer.), an alluvial gold-washing.

Lavra (Brazil), a small alluvial washing.

Le (China) = 486.176 yd.

Lead, a well-defined bed of pay-dirt.

Leadings (Australia), barren drift overlying pay-dirt.

League (Brazil) = $3\frac{1}{2}$ miles.

League (Spanish) = 20,000 Spanish ft. = 5572 mètres = 6093 yd.

Leang (China) = $1\frac{1}{4}$ oz. av.

Ley de oro (Mexico, &c.), properly the fineness of the gold, but apparently also applied to the assay value of an ore.

Lilin kalulut (Malay), a wax used by gold assayers.

Long-tom, see p. 162.

Mace (China) = 7*d*.

Manta (Nicaragua), a surface deposit of broken quartz, worked for gold.

Muraye (Argentine Rep.), grinding mortar used by the Indians for reducing quartz.

Marco or *Marc*, see Spanish goldsmiths' weights.

Mark (Austria), old weight, 0·618 lb. av.

Mas muda (Malay), "young" gold, or gold below 8 *mutu*.

Mas tuah (Malay), "old" gold, or gold from 8 to 10 *mutu*.

Mergulhar (Brazil), washing auriferous river-sands.

Miam (Straits Set.), a gold weight = 52 gr. ; 16 *miams* = 1 *bongkal*.

Milreis (Brazil) = 2*s*. 3*d*.

Miners' inch of water, see pp. 306-10.

Moco de hierro (Venezuela), a highly ferruginous rock, assuming the form of a conglomerate, a grit, a breccia, or even a pisolitic brown iron-ore ; it always consists mainly of limonite and earthy red hæmatite, with pebbles and angular fragments of quartz, schist, and felstone. In its decomposed form it is known as *tierra de flor* (q. v.).

Monton (Mexico), 17 *quintals* = 1560 lb.

Mullock (Australia), angular débris of the country rock filling a fissure.

Mundic, iron-pyrites.

Mutu (Malay), a term denoting the degrees of fineness of gold.

Napal, indurated white clay carrying auriferous quartz streaks in the Malay Peninsula.

Called also *steatite*.

Nariyas, gold-washers of the N.W. Provinces, India.

Nega (Thibet) = $\frac{1}{8}$ lb.

Nekozu, straw-mats specially woven, used for catching gold in the sluices in Japan.

Nugget, a sizable rounded piece of native gold.

Oiro (Port.), gold.

Oitavo (Brazil) = 55·34 gr. troy ; 8 *oitavas* = 1 *onça*.

Omorotchi (Russia), birch-bark skiffs.

Onça (Brazil) = 442·72 gr. troy.

Or (Fr.), gold.

Orang gulla (Sumatra), miners.

Oro (It., Span.), gold.

Oro corrido (U. S. Colombia), alluvial gold.

Ouro (Port.), gold.

Paint-gold, gold coating quartz pebbles in cement.

Palacurnæ (Latin), gold nuggets.

Palæ (Roman Spain), gold nuggets.

Palagæ (Latin), gold nuggets.

Pan, see p. 154.

Panella, see *Gamella*.

Paret (Borneo), a mine.

Pass (Heb.), gold.

Patach (Brazil) = 8·65*d*.

Patea, see *Batea*.

Pay-dirt, that portion of an alluvial deposit which contains gold in paying quantity.

Peña (U.S. Colombia), a more or less decomposed felspar or gneiss, coloured by iron and yielding gold as far as decomposition has taken place.

Pépita (Span. Amer.), a gold-grain.

Perjong, the Malayan crow-bar.

Peso (Mexico) = 4*s*.

Phdtang (N. Himálaya), a quantity of gold-dust melted into a lump, value about 8 rupees (16s.), and used as currency.

Phukpa (Thibet), miners' caves.

Piastre (Old Spanish) = 37·647*d*.

Picul (China) = 133½ lb. av.

Picul (Phillipines, &c.) = 139½ lb.

Picul (Siam) = 135½ lb.

Pie (India) = ½*d*.

Piedra morada, see *Porfiro*.

Piedra negra (Venezuela), greenish-grey felstone.

Pillah (Bokhara), gold.

Playa (Span. Amer.), a level bank.

Polvillos (México), tailings.

Pon (Tamil), gold.

Porfiro, or *porfido* (Venezuela), a reddish, pink, or brown ferruginous hornstone, sometimes becoming jasper, occasionally containing crystals of iron-pyrites, or cavities left by their decomposition, and rarely visible gold, but not in payable quantity. It occurs throughout the Caratal district, and is regarded as a favourable indication of gold. It is known also as *quartzo morado* and *pie-dra morada*.

Pozo (México), winze.

• *Pud* (Russia) = 36 lb. 2 oz. av.

Putti, a gold washing tray used in Madras.

Putty-stones (America), soft pieces of decomposed rock found in placer deposits.

Quartzo morado, see *Porfiro*.

Quebrada (Span. Amer.), a valley.

Quinto (Brazil), royalty on gold mining.

Race, an artificial water-course.

Rang (Ceylon), gold, whence Rang-galle (from *rang-welle*, golden sand).

Ratrang (Ceylon), melted gold.

Reefing, working auriferous reefs or veins.

Reef-wash, a deposit of wash-dirt spread over an expanse of flat or undulating bed-rock, or lodged in a hollow in bed-rock above the level of the gutter; an Australian term, apparently synonymous with bench (q. v.)

Rega (Brazil), a water conduit or launder.

Rei (Brazil) = ¼*d*.

Respaldo (México), wall of a lode.

Rifle or *ripple*, a groove about 1 in. deep at the lower side, diminishing towards the upper, with a width of about 3 in., cut across the riffle-board which forms a false bottom to a sluice, and partly filled with mercury while at work; also small strips of wood nailed across and rising above the floor of a box-sluice.

Rock, often applied to igneous rock-flows met with in deep leads.

Rock-bound (Australia), a reef not accompanied by gangue-stuff.

Ruble (Russia) = 3s. 6*d*.

Run, the duration of a spell of operations without stopping to clean up.

Rupée India) = 2s.

Saga (Straits Set.), a gold weight = 4½ gr.; 16 sagas = 1 *miam*.

Sana birro (W. Africa), gold nuggets.

Sana ku (W. Africa), gold-washing.

Sana manko (W. Africa), gold powder.

Sana mira (W. Africa), gold-rust (? emery).

Sarshoo (Thibet) = $\frac{3}{8}$ oz.

Scad (America), a term for a nugget, but not in general use.

Segullo (Spain), earth overlying auriferous deposits and indicative of gold.

Sepé (U. S. Colombia) a porphyroidal clay, stained with iron salts, sometimes covering the auriferous *peña*, and usually of very limited thickness.

Shaking-table, see p. 537.

Sickened mercury, see *Floured mercury*.

Sludge, mud flowing from a puddling-machine.

Sluice, see p. 169.

Sluice head, a definite measure of water drawn off for use.

Slum or *slimes*, the waste mud flowing from a quartz battery.

Sol (Latin), gold.

Soloto (Russ.), gold.

Spanish goldsmiths' weights.—The Spanish goldsmiths' lb. = 9600 *granos* of $\frac{800}{178}$ troy gr., thus divided :—

12 granos	= 1 tomin.	50 castellanos	= 1 marco.
8 tomines	= 1 castellano.	2 marcos	= 1 libra.

Spotted (America, &c.), leads in which the gold is irregularly disseminated.

Stamp, a solid head or weight used for crushing mineral.

Stamp-box, a box, generally of cast-iron, in which the stamps work, and whose sides are fitted with gratings.

Strake, see p. 516.

Strike, the longitudinal direction of a vein.

Suna (Duk., Guz., Hindi), gold.

Surfacing, working shallow auriferous alluvions.

Suvarna (Sanskrit), gold.

Tabah (Sumatra), a crow-bar used in gold-mining.

Tael (China) = $1\frac{1}{4}$ oz. av.

" " = 5s. 10d.

Tael (Straits Set.), a gold weight = 832·84 gr.

Tailings, the detrital mud flowing from gold-washing or crushing apparatus.

Tail-race, an aqueduct for conveying away dirty water and tailings.

Talutium (Latin), gold found in superficial deposits.

Tambang (Sumatra), mines.

Tambikiri quali (Malay), a black incrustation found on auriferous quartz.

Tan (China) = $133\frac{1}{4}$ lb. av.

Tapanhoacanga, see *Canga*.

Tharu, a gold-washing race in Champaran, Nepal.

Tial (Sumatra) = $610\frac{1}{2}$ gr.

Tibr (Arabic), gold dust.

Tical (Burma) = 252 gr.

" " = 2s. 6d.

Tical (Siam), for gold = 236 gr. troy, or nearly $\frac{1}{2}$ oz.

Tierra de flor (Venezuela), decomposed *moco de hierro* (q. v.) washed down the hill-sides, occupying areas of 100 to 200 acres and 6 to 10 ft. thick, and probably deriving its ferruginous matter from iron-pyrites. It is considered a good indication of alluvial gold.

Tile ore (Australia), ferruginous copper oxide.

Toise = 6·395 ft.

Tola (India), 180 gr.

Ton (Siam), gold.

Torpedo (Dutch Guiana), a sloping iron perforated plate fixed between the long-tom and the riffles in the sluice (p. 163).

Tosca (U. S. Colombia), a volcanic rock overlying the auriferous strata, and considered a good indication.

Tsuru, picks used for loosening the auriferous gravel for washing in Japan.

Underlie, dip.

Vara (Mexico, &c.) = 2.782 ft.

Venéro (U. S. Colombia), a vein or bed.

Volost (Russia), a commune of peasants.

Wash-dirt, the auriferous gravel, sand, clay, or cement in which the greatest proportion of gold is found.

Złoto (Polish), gold.

Złoty (Araucic), gold.

Złoto (Polish), gold.

Zolotnik (Russia) = 65.83 gr.

TABLE FOR CONVERTING PERCENTAGES INTO TROY WEIGHT PER STATUTE TON.

Percentage.	Per Ton.			Percentage.	Per Ton.		
	oz.	dwt.	gr.		oz.	dwt.	gr.
0.0001	0	0	15.68	0.06	19	12	0
0.0002	0	1	7.36	0.07	22	17	8
0.0003	0	1	23.04	0.08	26	2	16
0.0004	0	2	14.72	0.09	29	8	0
0.0005	0	3	6.40	0.1	32	13	8
0.0006	0	3	22.08	0.2	65	6	16
0.0007	0	4	13.76	0.3	98	0	0
0.0008	0	5	5.44	0.4	130	13	8
0.0009	0	5	21.12	0.5	163	6	16
0.001	0	6	12.8	0.6	196	0	0
0.002	0	13	1.6	0.7	228	13	8
0.003	0	19	14.4	0.8	261	6	16
0.004	1	6	3.2	0.9	294	0	0
0.005	1	12	16.0	1.0	326	13	8
0.006	1	19	4.8	2.0	653	6	16
0.007	2	5	17.6	3.0	980	0	0
0.008	2	12	6.4	4.0	1,306	13	8
0.009	2	18	19.2	5.0	1,633	6	16
0.01	3	5	8.0	6.0	1,960	0	0
0.02	6	10	16.0	7.0	2,286	13	8
0.03	9	16	0.0	8.0	2,613	6	16
0.04	13	1	8.0	9.0	2,940	0	0
0.05	16	6	16.0	10.0	3,266	13	8

Example.—500 grains of ore gave 0.044 grains gold, what is the yield per ton?

One hundred grains of the ore will give $0.044 \div 5 = 0.0088$ grains; and

Per cent. oz. dwt. gr.
According to table 0.008 = 2 12 6.4

„ „ 0.0008 = 0 5 5.44

So 0.0088 = 2 17 11.84 per ton.

GOLD RECKONER, BY WHICH THE VALUE OF ANY NUMBER OF OUNCES, PENNYWEIGHTS, AND GRAINS MAY BE READILY CALCULATED.

At per Ounce	Ounces.				Pennyweights.								Grains				At per Ounce								
	2		3		4		6		8		10		1		2			3		4					
	℥	s	d	℥	s	d	℥	s	d	℥	s	d	℥	s	d	℥		s	d	℥	s	d			
℥ s d	5	0	0	7	10	0	10	0	0	15	0	℥ <td>s<td>d</td><td>℥<td>s<td>d</td><td>℥<td>s<td>d</td><td>℥<td>s<td>d</td></td></td></td></td></td></td></td>	s <td>d</td> <td>℥<td>s<td>d</td><td>℥<td>s<td>d</td><td>℥<td>s<td>d</td></td></td></td></td></td></td>	d	℥ <td>s<td>d</td><td>℥<td>s<td>d</td><td>℥<td>s<td>d</td></td></td></td></td></td>	s <td>d</td> <td>℥<td>s<td>d</td><td>℥<td>s<td>d</td></td></td></td></td>	d	℥ <td>s<td>d</td><td>℥<td>s<td>d</td></td></td></td>	s <td>d</td> <td>℥<td>s<td>d</td></td></td>	d	℥ <td>s<td>d</td></td>	s <td>d</td>	d		
2 10 0	5	0	0	7	17	6	10	10	0	0	15	0	1	5	0	1	5	0	1	5	0	2	10	0	
2 12 6	5	5	0	8	5	0	10	6	0	15	9½	1	6	3	1	1	1	1	1	6	3	2	12	6	
2 15 0	5	10	0	8	5	0	11	0	0	0	16	0	1	7	0	1	7	0	1	7	0	2	15	0	
2 17 6	5	15	0	8	12	6	11	6	0	0	17	3	1	8	9	1	8	9	1	8	9	2	17	6	
3 0 0	6	0	0	9	0	0	12	0	0	0	18	0	1	10	0	1	10	0	1	10	0	3	0	0	
3 12 0	7	4	0	10	16	0	14	4½	0	1	1	7	1	16	0	1	16	0	1	16	0	3	12	0	
3 12 6	7	5	0	10	17	6	14	6	0	1	1	9	1	16	3	1	16	3	1	16	3	3	12	6	
3 13 0	7	6	0	10	19	0	14	7	0	1	1	10½	1	16	6	1	16	6	1	16	6	3	13	0	
3 15 0	7	10	0	11	5	0	15	0	0	1	2	6	1	17	6	1	17	6	1	17	6	3	15	0	
3 15 3	7	10	6	11	5	9	15	1	0	1	2	6½	1	17	7½	1	17	7½	1	17	7½	3	15	3	
3 15 6	7	11	0	11	6	0	15	1	2	0	1	2	7½	1	17	9	1	17	9	1	17	9	3	15	6
3 15 9	7	11	6	11	6	6	15	1	2	½	1	2	8½	1	17	10½	1	17	10½	1	17	10½	3	15	9
3 16 0	7	12	0	11	8	0	15	4	0	1	2	9½	1	18	0	1	18	0	1	18	0	3	16	0	
3 16 3	7	12	6	11	8	9	15	5	0	1	2	10½	1	18	1½	1	18	1½	1	18	1½	3	16	3	
3 16 6	7	13	0	11	9	0	15	6	0	1	2	11½	1	18	3	1	18	3	1	18	3	3	16	6	
3 16 9	7	13	6	11	9	6	15	3	0	1	2	11½	1	18	4½	1	18	4½	1	18	4½	3	16	9	
3 17 0	7	14	0	11	11	0	15	4½	0	1	3	1	1	18	6	1	18	6	1	18	6	3	17	0	
3 17 3	7	14	6	11	11	6	15	4½	0	1	3	1	1	18	6	1	18	6	1	18	6	3	17	3	
3 17 6	7	15	0	11	12	0	15	5	0	1	3	3	1	18	9	1	18	9	1	18	9	3	17	6	
3 17 10½	7	15	9	11	13	7½	15	10	0	1	3	3	1	18	9	1	18	9	1	18	9	3	17	6	
3 18 0	7	16	0	11	14	0	15	12	0	1	3	4½	1	19	0	1	19	0	1	19	0	3	18	0	
3 18 3	7	16	6	11	14	6	15	13	0	1	3	4½	1	19	1	1	19	1	1	19	1	3	18	3	
3 18 6	7	17	0	11	15	0	15	14	0	1	3	5½	1	19	3	1	19	3	1	19	3	3	18	6	
3 18 9	7	17	6	11	16	3	15	15	0	1	3	6½	1	19	4½	1	19	4½	1	19	4½	3	18	9	
3 19 0	7	18	0	11	17	0	15	16	0	1	3	7½	1	19	6	1	19	6	1	19	6	3	19	0	
3 19 3	7	18	6	11	17	6	15	17	0	1	3	8½	1	19	7	1	19	7	1	19	7	3	19	3	
3 19 6	7	19	0	11	18	0	15	18	0	1	3	9½	1	19	9	1	19	9	1	19	9	3	19	6	
3 19 9	7	19	6	11	19	3	15	19	0	1	3	10	1	19	10½	1	19	10½	1	19	10½	3	19	9	
4 0 0	8	0	0	12	0	0	16	0	0	1	4	0	1	2	0	2	0	0	2	0	0	4	0	0	

NOTE.—℥3 17s. 10½d. is the standard price for British gold.

TABLE SHOWING THE QUANTITY OF FINE GOLD IN ONE OUNCE OF ANY ALLOY TO ONE-EIGHTH OF A CARAT, AND THE MINT VALUE OF THE GOLD IN ONE OUNCE OF EACH ALLOY.

Fine Gold.			Carat Gold.			Sterling Value.		
oz.	dwt.	gr.	carats.	grs.	eighths.	£	s.	d.
1	0	0	24	0	0	4	4	11'454
0	19	4	23	0	0	4	1	4'977
0	18	8	22	0	0	3	17	10'500
0	17	12	21	0	0	3	14	4'023
0	16	16	20	0	0	3	10	9'545
0	15	20	19	0	0	3	7	3'068
0	15	0	18	0	0	3	3	8'590
0	14	4	17	0	0	3	0	2'113
0	13	8	16	0	0	2	16	7'636
0	12	12	15	0	0	2	13	1'159
0	11	16	14	0	0	2	9	6'682
0	10	20	13	0	0	2	6	0'204
0	10	0	12	0	0	2	2	5'727
0	9	4	11	0	0	1	18	11'250
0	8	8	10	0	0	1	15	4'773
0	7	12	9	0	0	1	11	10'295
0	6	16	8	0	0	1	8	3'818
0	5	20	7	0	0	1	4	9'341
0	5	0	6	0	0	1	1	2'863
0	4	4	5	0	0	0	17	8'386
0	3	8	4	0	0	0	14	1'909
0	2	12	3	0	0	0	10	7'432
0	1	16	2	0	0	0	7	0'954
0	0	20	1	0	0	0	3	6'477
0	0	15	0	3	0	0	2	7'858
0	0	10	0	2	0	0	1	9'239
0	0	5	0	1	0	0	0	10'619
0	0	4'375	0	0	7	0	0	9'292
0	0	3'750	0	0	6	0	0	7'964
0	0	3'125	0	0	5	0	0	6'637
0	0	2'500	0	0	4	0	0	5'309
0	0	1'875	0	0	3	0	0	3'982
0	0	1'250	0	0	2	0	0	2'655
0	0	0'625	0	0	1	0	0	1'327

								£	s.	d.
1 oz.	pure gold	is worth	..	..	..	..	..	4	4	11
1 dwt.	"	"	..	..	..	..	..	0	4	3
1 gr.	"	"	..	..	..	..	..	0	0	2

Example.—If a sample of bullion is 20 carats 3 grs. 5 eighths, how many troy dwts. and grs. are there in it, and what is the value of the gold in the alloy?

According to table—

								£	s.	d.
20 carats	..	..	..	=	16	16	..	=	3	10 9'545
3 grains	..	..	..	=	0	15	..	=	0	2 7'858
5 eighths	..	..	..	=	0	3'125	..	=	0	0 6'637
Answer	..	..	..		17	10'125			3	14 0'040

CHAPTER XII.

SUPPLEMENTARY.

The matter contained in this Chapter was received too late for insertion in its proper place.

Jordan's Patent Process.—The machinery comprising this process is intended by the inventor as a substitute for the well-known Californian stamp batteries and ordinary amalgamated plates, which have been fully described in the earlier pages of this work.

The points claimed in its favour, as compared with the ordinary gold mill, are well worth considering, and will, no doubt receive the serious attention of those interested, because it has long been felt, more especially since the opening up of the extensive gold fields of South Africa, that the old-fashioned stamp battery and amalgamation appliances are not calculated to meet the requirements of the age, and the various conditions under which it is necessary to treat gold-bearing ores.

The drawbacks to stamps and the ordinary process of extracting the gold by means of amalgamated plates, may be enumerated as follows :—

1st. The excess of power employed in the reduction of the quartz, consequent upon the intermittent and uncertain action of the stamp battery.

2nd. The small proportion of work done for a given weight and cost of machinery employed.

3rd. Excessive wear and tear owing to the quantity of plant necessary and the number of wearing surfaces and details consequent upon its design and character.

4th. The costly nature of the heavy foundations necessary for the proper erection of the machinery.

5th. The enormous cost of freight and transport to the majority of gold-bearing districts, and in many cases the impossibility of conveying such plant to the required site, unless constructed in sections so as to materially interfere with the stability and permanent value of the battery.

6th. The unsatisfactory and uncertain results derived from the use of amalgamated copper plates in their present form, the amount of skilled

attention they require to keep them in working order, and the fact that it is almost impossible to protect the large exposed surface covered by the amalgam.

It is to overcome these objections as far as possible that Mr. Jordan has invented the two machines which, with an ordinary stonebreaker, comprise his process.

● It is not proposed to give a detailed account of this new machinery, as some of the foreign patents are, at the time of publication, still not fully secured. It may, however, be stated that the reducing machinery which takes the place of the stamp is said to be less than one-third the weight; is far more simple in design and construction; is continuous and automatic in its action; and capable of reducing a given quantity of material at the expense of less than two-thirds of the power.

The action of the amalgamator is said to ensure absolute contact between its amalgamated surfaces and the lightest particle of gold, which is the one thing necessary to prevent the great percentage of loss in dealing with ore carrying fine or "float" gold.

This machine is also perfectly simple and automatic, requiring little or no attention when working. What would seem to be another important feature in this amalgamator is, that the amalgam is enclosed in the machine and can be kept under lock and key.

An installation of the plant is to be erected in London, for exhibition under practical working conditions.

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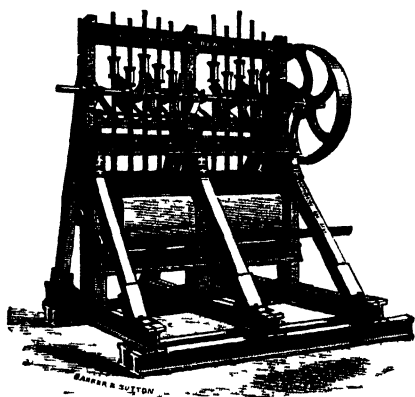
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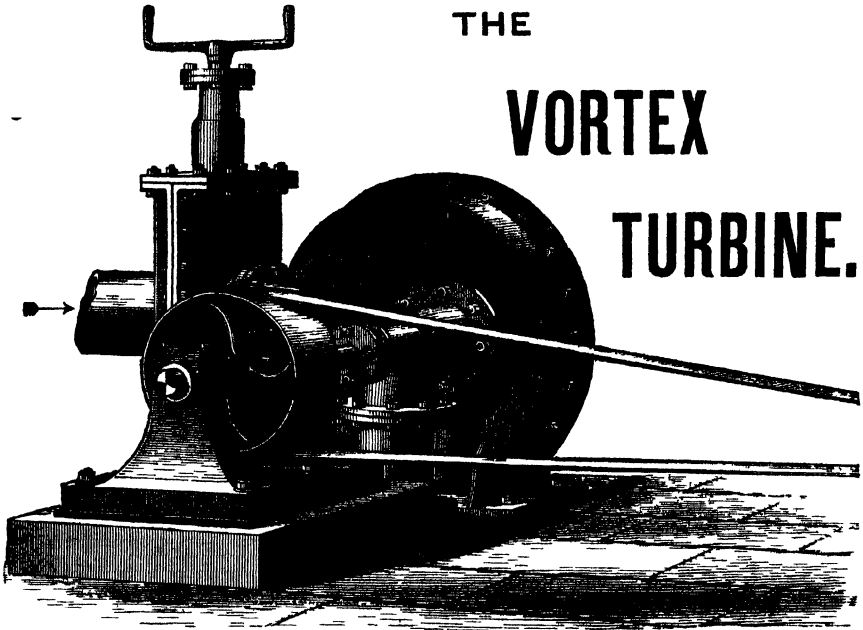
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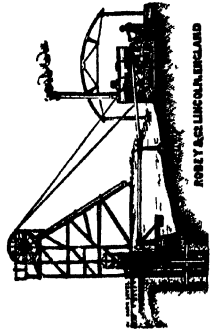
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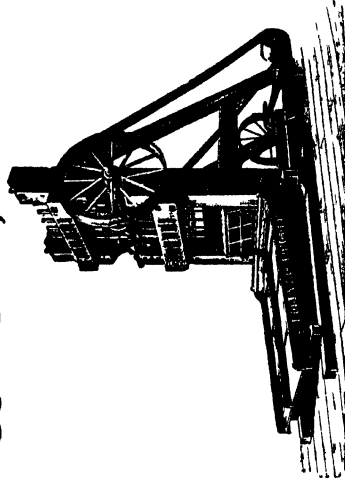
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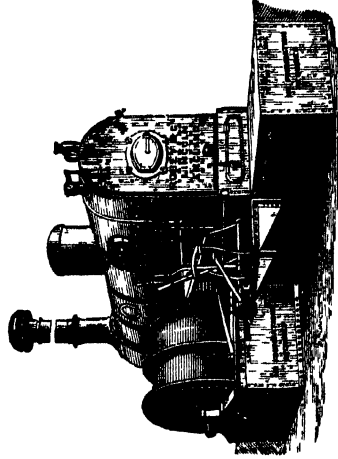
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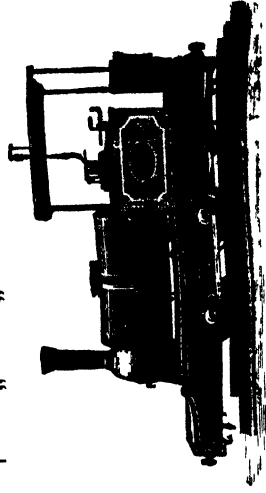
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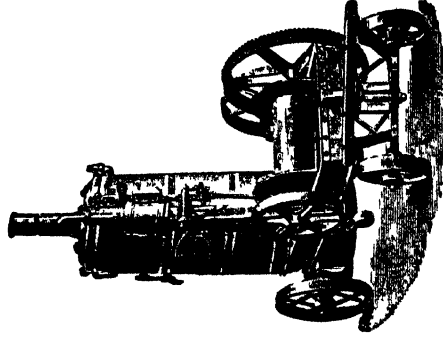


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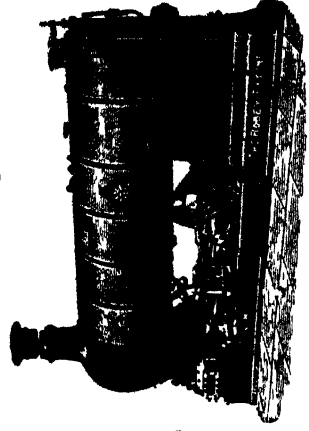


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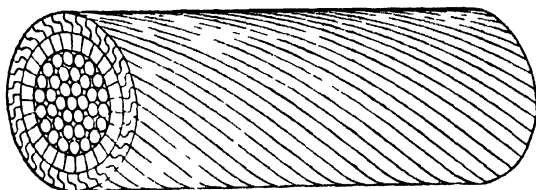
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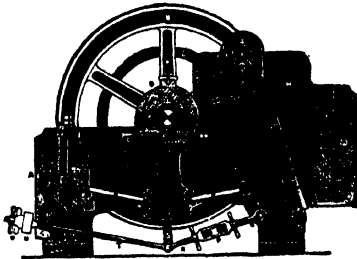
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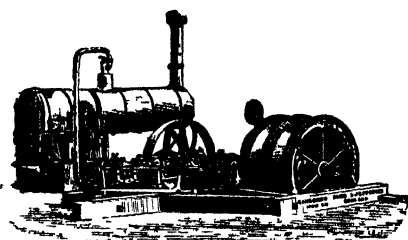
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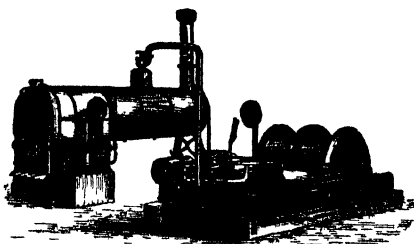
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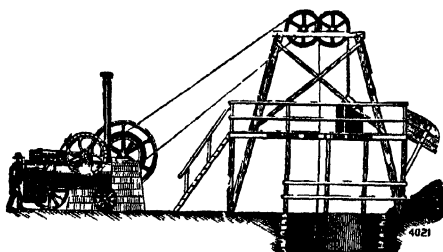


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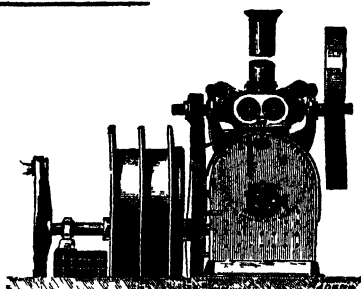


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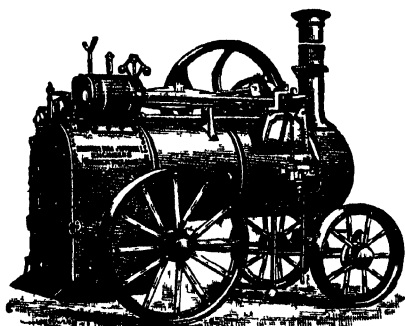


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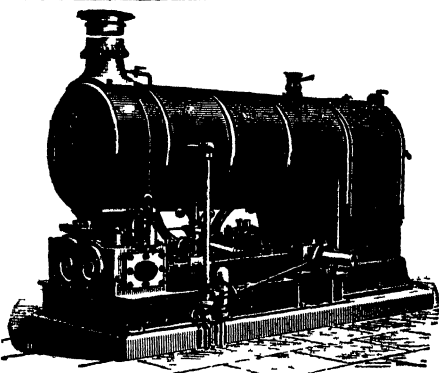


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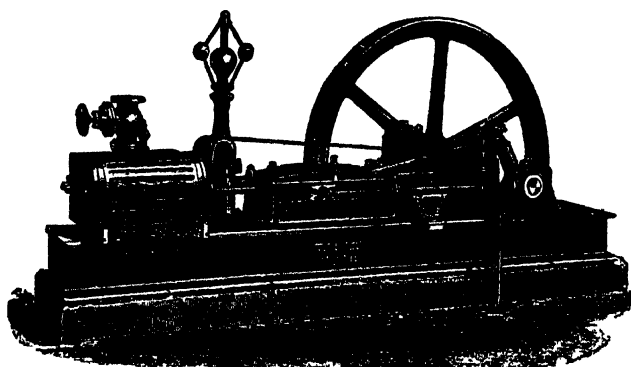
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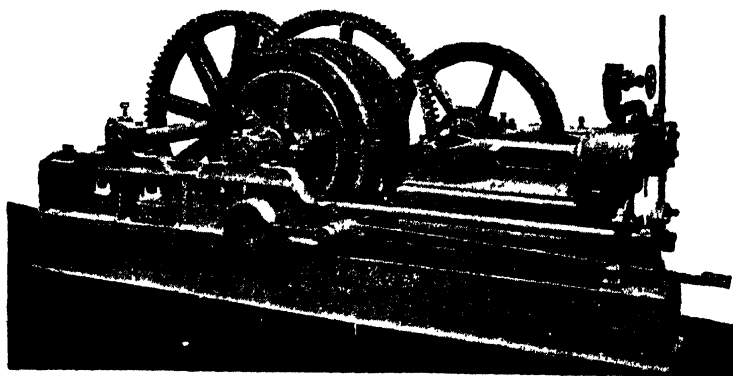
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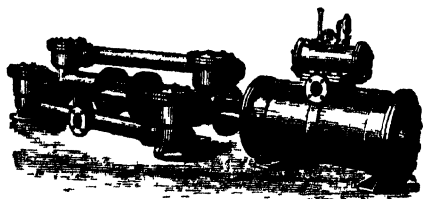
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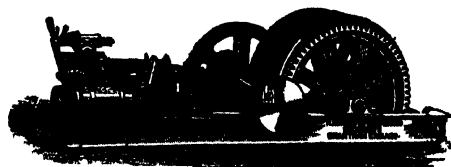
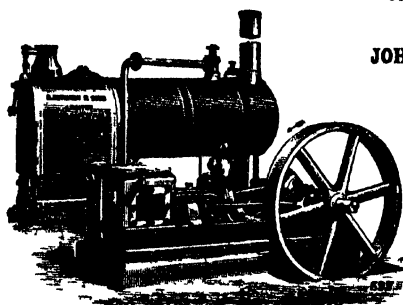
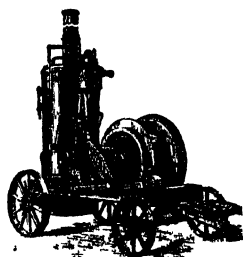
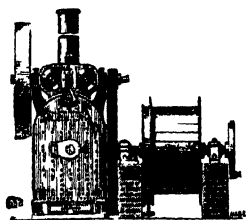
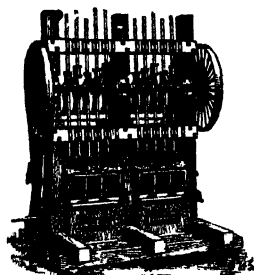
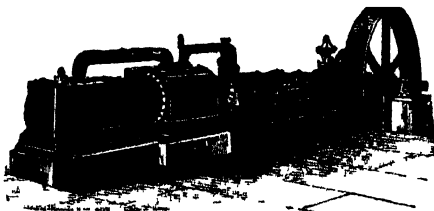
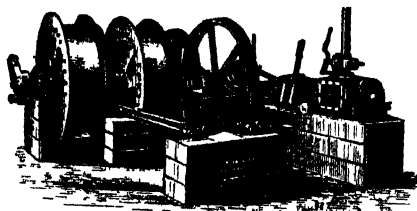


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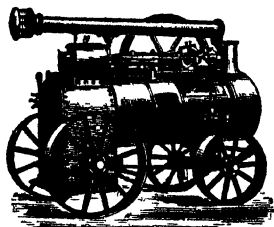
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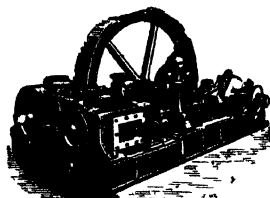
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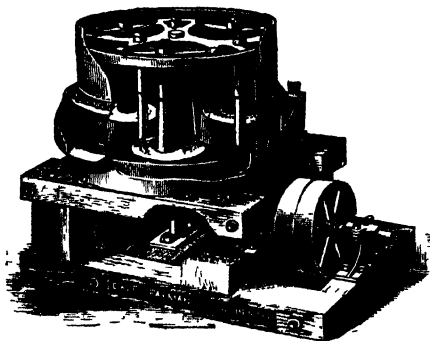
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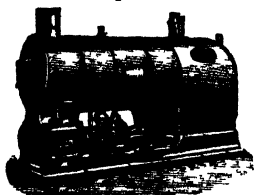


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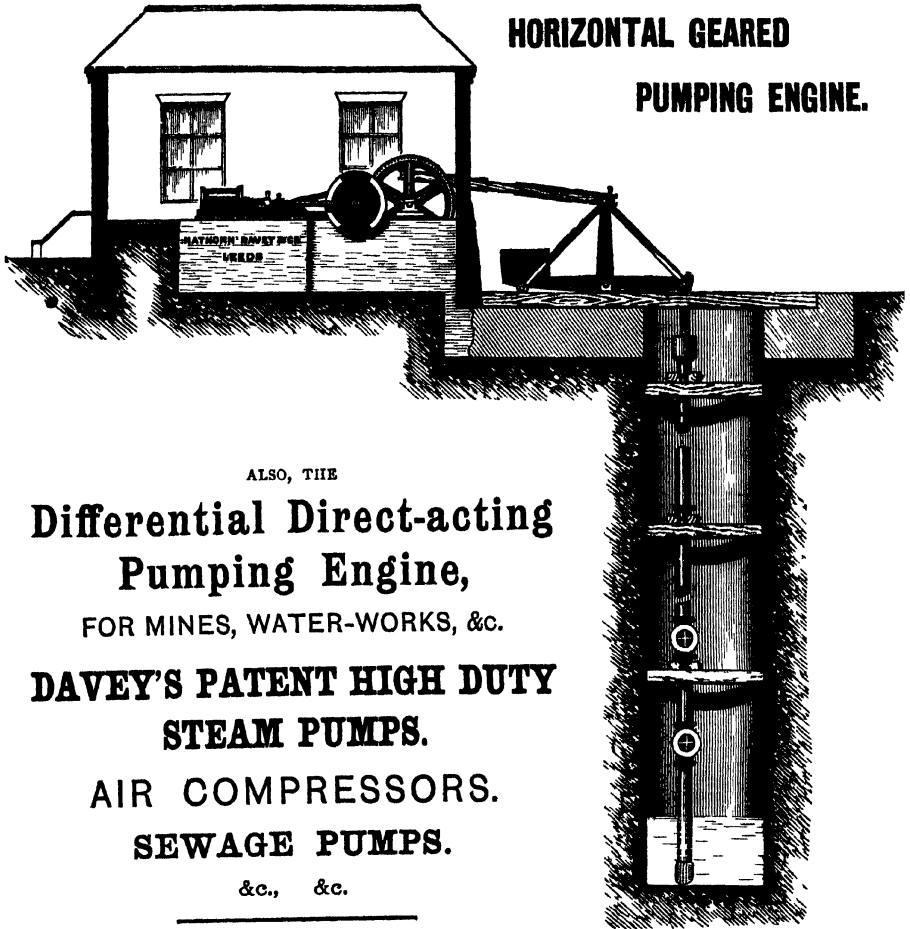
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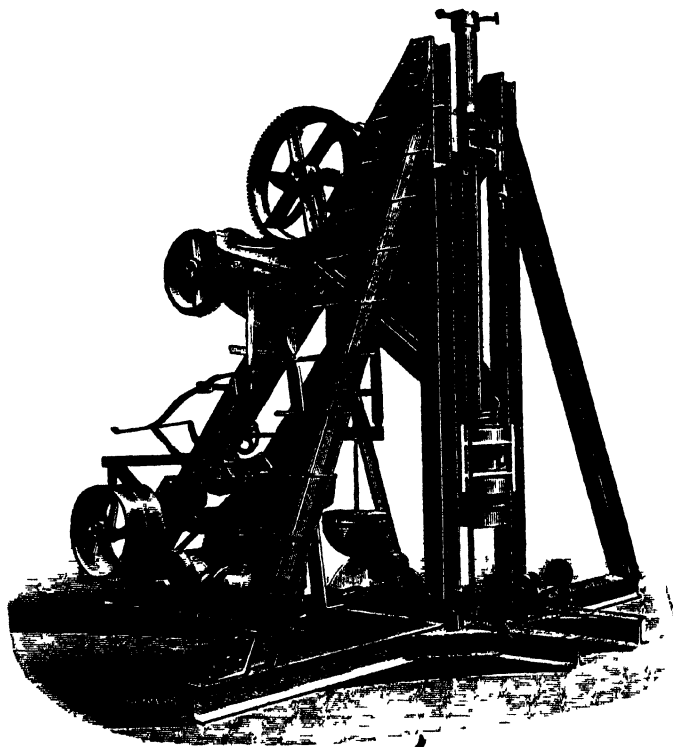
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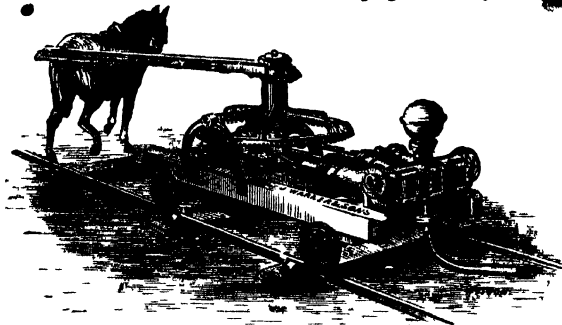
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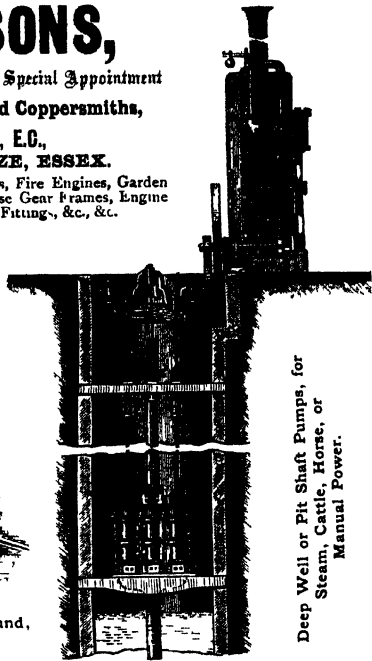
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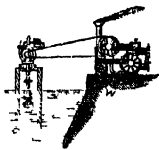


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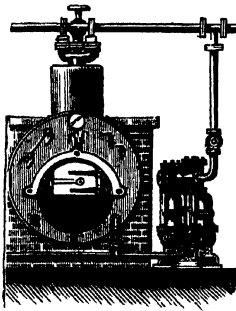
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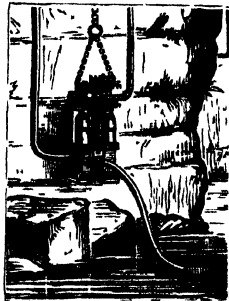
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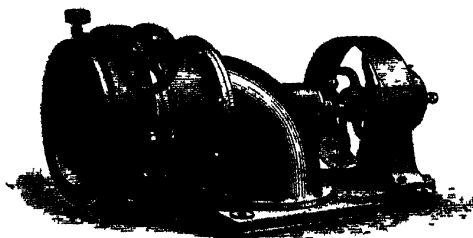
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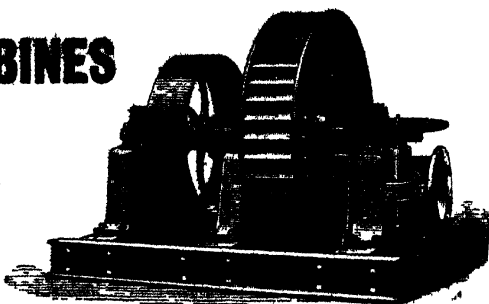
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